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COAL AND COKE

PREFACE

The preparation of this book was undertaken in part because of the requests for information made to the author by numerous friends engaged in the manufacture of coal gas, and in part with the desire of placing before the American student of coal gas production, data in a concise form covering the various systems of coal carbonization.

Baltimore was the first city in America to undertake to supply illuminating gas as a commercial commodity. The residents of this city were invited in 1816 to visit Peale's Museum to see the new "burning air," the name given to coal gas by the advertising agent of the museum. This exhibition took place just six months after the daily papers had announced that Covent Garden Theatre in London, as well as several streets in the English capitol, were illuminated by the burning of coal gas.

Unfortunately, coal gas manufacture received a serious setback through the advent of carburetted water-gas, and this condition has continued, until now approximately 75 per cent. of the gas manufactured is produced in the water-gas generator.

This condition is the reverse of that existing in England, which country can truly be termed the home of coal gas, and consequently the successful installation of coal gas plants in America will necessarily have to follow English practice.

English pioneers and investigators, such as the late Vivian B. Lewes, and many others, have paved the way for the revival of coal gas manufacture in this country, and there is evidence that such a revival is about due. The success of this development will depend in large measure upon the use of the great fund of knowledge covering the carbonization of coal, which has been produced by the indefatigable labors of the English pioneers and investigators.

Much information having direct bearing on the carbonization of coal is being made available by the U. S. Bureau of Mines, and its bulletins are of extreme value to the students of this subject, while the investigations undertaken by the University of Illinois are opening up new fields of research which will ultimately yield a better understanding of the subject.

I wish to express my indebtedness to the various journals from which information has been drawn, and to the authors of articles whose names are given in these pages, for the fund of knowledge placed at my disposal. I would appreciate suggestions from the readers of this book, which may lead to an extension of its usefulness in later editions.

FREDERICK H. WAGNER.

BALTIMORE, MD.,
January, 1916.

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COAL AND COKE

PART I. COAL

CHAPTER I

ORIGIN AND CLASSIFICATION

All coals, no matter what their classification, are of purely vegetable origin, generally attributed to the remains and débris of prehistoric forests, these remains having been subjected to a species of destructive distillation, due to varying, but great, temperatures and pressures, and the original organic matter has thus been resolved into its ultimate constituents, such as carbon, hydrogen, oxygen and other substances, all in varying quantities, due to the method of formation.

The formation of coal was greatly influenced by time, location of the bed beneath the surface, and disturbances due to geologic upheavals, the latter resulting in the admixture of various mineral substances with the final production of the various classifications, such as Peat, Lignite, Cannel, Bituminous, Semi-bituminous, and Anthracite Coals, the classification of the coals being generally accepted under the specifications given in Table I.

Table I

Classification	Per cent. of combustible		B.t.u. per pound of combustible
	Fixed carbon	Volatile matter	
Anthracite	97.0 to 92.5	3.0 to 7.5	14,600 to 14,800
Semi-anthracite	92.5 to 87.5	7.5 to 12.5	14,700 to 15,500
Semi-bituminous	87.5 to 75.0	12.5 to 25.0	15,500 to 16,000
Bituminous, eastern	75.0 to 60.0	25.0 to 40.0	14,800 to 15,300
Bituminous, western	65.0 to 50.0	35.0 to 50.0	13,500 to 14,800
Lignite	Under 50	Over 50	11,000 to 13,500

The uncombined carbon in coal is generally termed "fixed carbon," and as some of the carbon is found in combination with hydrogen which, together with other gases, is driven off through the application of heat, the volatile matter is formed by this process of distillation, the fixed carbon together with the volatile matter forming what is known as the "combustible" constituent of the coal. This volatile matter, however, also contains nitrogen and oxygen, which are not combustible, but as custom has applied this term to that portion of the coal which is free from ash and dry, nitrogen and oxygen are generally included under this term. Coal also contains moisture and refractory earthy materials, the latter forming the ash. A variation of from 3 to 30 per cent. of the total weight of the coal is found in the ash, while the moisture may vary from 0.75 to 45 per cent. of this weight, depending upon the district where the coal is mined as well as upon the grade of the coal.

The gradation in the formation of coal from its original state is best shown by the tabular statement in Table II, where the percentage of carbon, hydrogen, and oxygen is given for the various periods of formation.

TABLE II

Substance	Carbon	Hydrogen	Oxygen
Wood fiber	52.65	5.25	42.10
Peat	59.57	5.96	34.47
Lignite	66.04	5.27	28.69
Earthy brown coal	73.18	5.68	21.14
Bituminous coal	75.06	5.84	19.10
Semi-bituminous coal	89.29	5.05	5.66
Anthracite coal	91.58	3.96	4.46

Various equations have been written to show the chemical changes undergone by the original vegetable matter in the formation of coal, but they are all based upon assumptions which cannot be proven. These equations are usually based upon the hypothesis that the vegetable matter is cellulose, a carbohydrate with the formula $C_6H_{10}O_5$, and upon the assumption that this carbohydrate gradually gives up carbon dioxide, marsh gas, and water, thus gradually producing the series of matter commonly known as coal. The formula $C_6H_{10}O_5$ represents the empirical composition of cellulose, but not its true molecular weight, the formula given by A. NASTUKOFF being 40 $C_6H_{10}O_5$, but this also is an assumption.

Cellulose is a very important constituent of vegetable matter, and it is true that during decay it gives up carbon dioxide, march gas, water, and other matters, but these changes are difficult to determine. The waters in every swamp or peat bog contain soluble organic matter, and the usual equations given for the formation of coal do not take this into account, but its presence is a factor in the resultant product.

If the composition of wood is used as the basis of coal formation, the derivatives can be better traced, see Table II, but here again wood, or the fibers of wood, the ultimate composition of which has been definitely determined, is not the only substance which must be considered in arriving at a definite conclusion. This substance contains cellulose, but also another substance known as lignone, lignin, or lignocellulose, as well as other organic matters.

In tracing the actual formation of coal it is safe to assume that the original vegetation died and was subjected to a partial decay, this decaying matter being buried under successive layers of like matter, or of earthy sediments. This process prevented complete oxidation by contact with the atmosphere, and also subjected the matter to the heat of fermentation, as well as to a gradually increasing pressure. This vegetable origin can be easily traced by a microscopic examination of the final product, where the remains of trees, ferns, grass, sedge, moss, etc., are easily discerned, and thus the slow decay of this buried matter forms the chief interest in chemical investigations of the formation of coal; it must also be remembered, however, that the remains of animals also formed a portion of these substances, thus increasing the nitrogen content of the coal, while the ash due to these remains was increased by the addition of inorganic sediment.

Peat.—It is a generally accepted fact that the formation of peat is the first step in the formation of coal, due to the burial of mosses, grasses, and such plants as grow in marshes, in successive layers, this phenomena being easily traced by observation only, as the surface of a peat bog shows growing plants, while somewhat below the surface the remains of these plants can be seen; going a little deeper, a black substance, which no longer shows the vegetable origin to a marked degree, is found, this substance being known as peat.

Peat, then, is organic matter in the first stages leading to the formation of coal; its moisture content is very high, averaging as much as 75 or 80 per cent., while its ash content varies from 3 to 12 per cent.

Table III gives a series of analyses made by J. **Websky**,[1] the peat samples being dried at 100°, and the analyses calculated on an ash-free basis.

Table III.—Analyses of Sphagnum and Peat

	A	*B*	*C*	*D*	*E*	*F*	*G*
C.......	49.88	50.33	50.86	59.71	59.70	59.71	62.54
H.......	6.54	5.99	5.80	5.27	5.70	5.27	6.81
O.......	42.42	42.63	42.57	32.07	33.04	32.07	29.24
N.......	1.16	1.05	0.77	2.95	1.56	2.95	1.41
	100.00	100.00	100.00	100.00	100.00	100.00	100.00

A. Sphagnum, the chief plant of peat bogs. *B*. Light peat, near surface. *C*. Light peat. *D*. Moderately light peat. *E*, *F*. Black peat. *G*. Heavy brown peat.

Here the progressive increase in carbon while passing from sphagnum to heavy peat is clearly shown. The nature of the changes during the transformation of sphagnum, or vegetable matter, into peat are only partly understood, but it is a well-known fact that when ligneous fibers decay they yield an amorphous mixture of such substances known as humus acids, humic, ulmic, crenic, and apocrenic acids; these acids are not very well defined, and various formulæ have been given them, but none are accepted as established. When these acids are placed in an alkaline solution, they dissolve and are partly washed away, but the salts which are formed in combination with lime and iron, being insoluble, remain behind. Thus the ash of peat is rich in lime and iron, but the lime does not exist as carbonate.

It is presumed that the formation of humus acids is produced by a process of fermentation, thus setting free and eliminating some carbon, hydrogen, and oxygen in the shape of carbon dioxide, marsh gas, and water, these changes being probably assisted by the presence of micro-organisms. H. **Bornträger** made a series of peat analyses in which he found from 25 to 60 per cent. of humic substances, and from 30 to 60 per cent. of fiber in the black varieties, two of these analyses being given in Table IV

---

[1] Journal für praktische Chemie, Vol. 92, page 65, and The Data of Geochemistry, 1908.

TABLE IV

	A	*B*
Water	29.50	20.0
Ash	3.05	3.0
Fiber	54.95	47.0
Humus acids	12.50	30.0
	100.00	100.0

A. Light-colored peat, Hannover. Mean of two analyses. *B*. Black peat, Oldenburg.

This table shows that the ultimate changes have not proceeded to the same degree in light-colored peat as in the black peat.

Lignite.—Lignite is organic matter in the earlier stages of coal formation, and it includes all varieties of matter which are intermediate between peat and the coal of older formations, or between peat on the one side and bituminous coal on the other. This substance is also known as "brown coal," but the names lignite and brown coal are not always appropriate, as some lignites are not ligniform, and some brown coals are black in color. Geologically described, the lignites in composition bear a distinct relation to their age, as it is found that the younger variety closely approaches peat, while the oldest variety resembles true coal in its characteristics; while this statement is generally accepted, it does not show the true relation of the lignites, because cases are known where the lignites have been transformed into apparently bituminous coals, or even into anthracite. Lignites exist in several distinct varieties, such as:

1. True, or xyloid lignite, which is principally fossil wood and in which the ligneous structure is almost always found perfectly preserved.
2. Earthy brown coal, which is very earthy in structure and often contains fossil hydrocarbons or mineral resins.
3. Common brown coal, which is a compact form of lignite.
4. Pitch coal, which is also very compact, and receives its name for its peculiar luster.
5. Glance coal, another hard and very compact lignite, closely resembling the carboniferous coals.

6. Jet, which is the hardest variety of lignite, its derivation being probably due to the fossilization of coniferous wood, it being principally used for the production of jewelry and other ornaments.

Lignites usually contain a large amount of moisture, as they are peculiarly hygroscopic; they also contain organic sulphur as well as inclosures of pyrite or marcasite, and sulphates or gypsum. Table V gives the analyses of American lignites made at the fuel testing plant of the United States Geological Survey, all of the samples analyzed having first been air dried.

TABLE V.—ANALYSES OF AMERICAN LIGNITES

	A	*B*	*C*	*D*	*E*
Moisture	16.70	22.48	18.51	17.69	9.05
Volatile matter	37.10	31.36	35.33	37.96	36.70
Fixed carbon	39.49	26.73	30.67	39.56	43.03
Ash	6.71	19.43	15.49	4.79	11.22
	100.00	100.00	100.00	100.00	100.00
Sulphur	0.63	0.56	3.05	0.63	1.76

A. Brown lignite, from Williston, North Dakota. *B*. Lignite from Texas. *C*. Lignite from the Tesla mine, Alameda County, Cal. *D*. Lignite from Wyoming. *E*. Black lignite from Red Lodge, Montana, which is a coal of doubtful character and is not really considered a lignite.

These coals were also analyzed as to their elements, ash, moisture, and sulphur being thrown out, the result, given in Table VI, showing less variation than the above.

TABLE VI.—ELEMENTARY ANALYSES OF AMERICAN LIGNITES OF TABLE V

	A	*B*	*C*	*D*	*E*
Carbon	72.62	73.63	75.19	75.97	77.47
Hydrogen	4.93	5.07	6.18	5.36	5.44
Nitrogen	1.20	1.35	1.04	1.41	1.75
Oxygen	21.25	19.95	17.59	17.26	15.34
	100.00	100.00	100.00	100.00	100.00

We find that the brown coals contain a great deal of the resinoids and fossil hydrocarbons in visible masses, as well as in a disseminated condition, and that organic solvents, such as benzene, will extract this matter from the lignites, but the resultant substances are not of a very definite composition, WATSON SMITH[1] having found 9.5 per cent. of substance soluble in benzene in a Japanese lignite, but this is a rare occurrence.

The lignites resemble peat more than carboniferous coals in their behavior in the presence of reagents, and they contain humic compounds which are soluble in caustic alkali solutions, but E. FREMY[2] states that xyloid lignite yields little, and compact lignite almost no ulmic acid in contact with alkaline solvents, FREMY also stating that lignite is soluble in alkaline hypochlorites, while the true coals are not. Lignites are also very sensitive to nitric acid, this admixture resulting in a yellow resinous body which is soluble in an excess of this reagent or in alkali solutions.

E. DONATH concluded that lignite is very dissimilar to true coal both chemically and as to origin, as the results in the presence of the reagents differ greatly, and they yield quite different products when subjected to destructive distillation; also that neither time nor heat can transform lignite into true coal, basing this hypothesis on the statement that lignite is derived from materials rich in lignocellulose, as shown by the presence of humic compounds, while the true coals were formed from substances which were fully, or nearly so, free from woody fiber, and the proteids of animal matter assisted in the formation of bituminous coal.

While DONATH's statements may not be universally accepted, his experiments seem to be substantiated by other investigators who have shown the distinct chemical differences between the lignites and the true coals: in the lignites humic compounds are readily found, but they are not so readily detected in the true coals; again, nitric acid reacts quickly on lignite, but the vigor of the reaction is much less on the true coals; it is, therefore, difficult to determine to what degree the true coals were derived from the lignites. Generally speaking, the specific gravity of lignite is low, varying from 1.2 to 1.23, and its appearance shades from a light brown to black, the former color exhibiting a distinct woody structure in the poorer varieties, while the black variety has a pitchy luster, such as hard coal, in the better grades. Lignite is non-caking and it burns with a bright but slightly smoky flame. It breaks very easily, and will therefore not stand much handling in transportation, while its exposure to the weather causes rapid disintegration, thus decreasing its burning possibilities.

---

[1] Journal Soc. Chem. Ind., 1891, page 975.

[2] Data of Geochemistry, 1908, page 651, F. W. CLARKE.

Bituminous Coal.—The composition of this coal shows that it occupies a position between the lignites and anthracite, but the exact line of demarcation is hard to recognize, as they at times overlap the brown coals. This coal is divided into two classes, semi-bituminous and bituminous, the former being softer than anthracite, and as it contains more volatile hydrocarbons, it kindles more readily and burns rapidly. The so-called bituminous coal is still softer and contains still more volatile hydrocarbons, the difference between the two grades being very important in an economical sense. The semi-bituminous variety has an average heating value per pound of combustible about 6 per cent. higher than the bituminous, and the former burns with less smoke than the latter, the distinctive characteristic of both varieties being the emission of a yellow flame and smoke when burning. In color the bituminous coals vary from dark brown to pitch black, the compact specimens usually exhibiting a resinous luster, while those grades which show traces of vegetable fiber usually have a silky luster.

Bituminous coals are of both the caking and non-caking variety, the former fusing and swelling in size when heated, the latter burning freely and not fusing under this condition, and being known as free-burning coals. The bituminous caking coals are rich in volatile hydrocarbons, and are therefore very valuable in the manufacture of coal gases. This coal readily absorbs atmospheric moisture, the surface moisture being easily removed by ordinary drying, but a portion of the water can be removed only by applying heat to the coal, this moisture being expelled at a temperature of about 250°F. (121°C.).

Another variety of bituminous coal is known as "Cannel" coal, and it is very rich in hydrogen and hydrocarbons, thus making it of extreme value in gas manufacture. This variety possesses a dull resinous luster and burns with an exceedingly bright flame without fusing, its approximate composition being: fixed carbon 25 to 55 per cent.; volatile matter 42 to 64 per cent., earthy matter 2 to 14 per cent. Its specific gravity is approximately 1.24.

Table VII gives the analyses of bituminous coal, of the Carboniferous age, as made at the fuel-testing plant of the United States Geological Survey, the proximate conventional analyses being:

TABLE VII.—PROXIMATE ANALYSES OF BITUMINOUS COAL

	A	*B*	*C*	*D*	*E*	*F*
Moisture	3.51	2.61	9.55	4.52	9.90	13.72
Volatile matter	16.82	34.92	36.19	40.96	33.66	36.24
Fixed carbon	73.04	56.30	43.65	38.99	44.86	39.72
Ash	6.63	6.17	10.61	15.53	11.58	10.32
	100.00	100.00	100.00	100.00	100.00	100.00
Sulphur	0.94	1.26	3.72	6.83	1.81	3.96

A. Ehrenfeld, Pennsylvania. *B*. Bruce, Pennsylvania. *C*. Vigo County, Indiana. *D*. Altoona, Iowa. *E*. Shawnee, Ohio. *F*. Staunton, Illinois.

The ultimate analyses of these same coals, on a water-, ash-, and sulphur-free basis are shown in Table VIII.

TABLE VIII.—ULTIMATE ANALYSES OF BITUMINOUS COALS

	A	*B*	*C*	*D*	*E*	*F*
Carbon	90.78	85.73	84.19	82.92	82.20	81.87
Hydrogen	4.69	5.49	5.92	6.06	5.45	5.85
Nitrogen	1.40	1.75	1.42	1.27	1.60	1.36
Oxygen	3.13	7.03	8.57	9.75	10.75	10.92
	100.00	100.00	100.00	100.00	100.00	100.00

Here the variation between carbon and oxygen is readily shown, the oxygen content increasing as the carbon content decreases.

The same content of constituents cannot be expected even in coals taken from the same mines, as is shown in Table IX, the latter being the analyses of samples taken from the Unser Fritz mine, Westphalia, the data having been prepared by F. FISCHER,[1] the last column of the table being the average of the series, prepared by CLARKE.

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[1] Data of Geochemistry, 1908; Zeitschrift für Angew. Chemie, 1894, page 605, and Technologie der Brennstoffe, Vol. 1, pages 518–520.

TABLE IX.—ANALYSES OF COAL FROM UNSER FRITZ MINE

Carbon	85.33	85.06	84.28	82.34	80.69	83.81
Hydrogen	5.20	4.66	4.85	4.94	4.94	4.98
Nitrogen	1.49	1.35	1.87	1.18	1.29	1.47
Oxygen	7.98	8.93	9.00	11.54	13.08	9.74
	100.00	100.00	100.00	100.00	100.00	100.00

Table X gives the analyses of fossil plants and cannel coal, these being peculiar variations of bituminous coals.

TABLE X

	A	*B*	*C*
Carbon	82.45	83.58	87.89
Hydrogen	4.75	5.77	6.53
Nitrogen	0.43	2.21	2.08
Oxygen	12.37	8.44	3.50
	100.00	100.00	100.00

A. An average of six analyses of fossil plants from the coal beds of Commentry, France, made by S. MEUNIR; these plants were in perfect structural preservation but had been entirely transformed into coal. See FREMY'S *Encyclopédie Chemique*, Vol. 2 (Complément, Part 1), page 152. *B*. Analysis of Wigan cannel, given by F. VAUX, Journal Chem. Soc., Vol. 1, page 320, 1840. *C*. Analysis of Tyneside cannel, given by H. TAYLOR, Edinburgh New Phil. Journal, Vol. 50, page 145, 1851. All three analyses are given on an ash-free basis.[1]

The variation in the nitrogen content in the above samples is very suggestive, as it shows that this content was very low in the fossil plants, but quite the reverse in the cannels; this seems to infer that the remains of the plants contributed very little nitrogen to the coal, and that the probable source of this constituent was due to the remains of animal matter in the cannels. It has also been pointed out by NEWBERRY that the remains of fish are abundantly found in cannel coal, seeming to infer that these cannel beds were deposited under water, the vegetable matter forming a carbonaceous paste in which the remains of fish became imbedded, this consolidation producing the cannel coal.

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[1] See The Data of Geochemistry, 1908, page 654.

Table XI[1] gives a comparison of several varieties of coals, moisture-, sulphur-, and ash-free, except where noted.

TABLE XI.—AVERAGE ANALYSES OF BITUMINOUS COAL

	A	*B*	*C*	*D*
Carbon	87.52	82.91	83.65	84.19
Hydrogen	5.20	5.70	5.48	5.58
Nitrogen	1.61	1.49	1.86	1.41
Oxygen	5.67	9.90	9.01	8.82
	100.00	100.00	100.00	100.00

A. Average of 20 analyses of bituminous coals from Pennsylvania, Maryland, Virginia, and West Virginia. Combined from data given in the report of the fuel-testing plant of the United States Geological Survey. *B*. Average of 40 analyses of bituminous coals from Ohio, Indiana, Illinois, Iowa, and Missouri. From the same report. *C*. Average of 15 analyses of Scotch coals, by W. D. ANDERSON and J. ROBERTS, Jour. Soc. Chem. Ind., Vol. 17, page 1013, 1898. Sulphur is included in the figure for oxygen. *D*. Average of 18 analyses of coals from Newcastle, 28 from Lancashire, and 7 from Derbyshire, England. Recalculated from averages cited by FISCHER in Chem. Tech. der Brennstoffe, Vol. 1, page 512.

Anthracite.—Anthracite, or hard coals, are those dry coals which contain from 3 to 7 per cent. of volatile matter and which do not fuse or swell when burned. True anthracite is a hard, compact, lustrous and sometimes iridescent coal, and it is characterized by but few joints and clefts. Its specific gravity varies between 1.4 to 1.8; it kindles with difficulty, is hard to keep burning, its flame being short and colorless, and it emits no, or very little, smoke. The semi-anthracite variety has less density, hardness and luster than true anthracite, its distinguishing characteristics, when newly fractured, being its ability to soot the hands; ordinarily its specific gravity is about 1.4, and it kindles more readily and burns more freely than true anthracite.

An analysis of anthracite shows a large proportion of fixed carbon, but a relatively small amount of volatilizable matter, as shown in Table XII.[2]

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[1] The Data of Geochemistry, 1908, page 654.

[2] Data of Geochemistry, 1908, page 656.

TABLE XII.—PROXIMATE ANALYSES OF ANTHRACITE

	A	*B*	*C*	*D*	*E*
Moisture	1.28	2.08	2.27	2.98	1.82
Volatile	12.82	7.27	8.83	3.38	6.18
Fixed carbon	73.69	74.32	78.83	87.13	86.75
Sulphur	....	....	0.68	0.66	0.75
Ash	12.21	16.33	9.39	5.85	4.50
	100.00	100.00	100.00	100.00	100.00

A. Semi-anthracite, Coal Hill, Arkansas. Prof. Paper, U. S. Geol. Survey, No. 48, 1906, page 202. *B*. Anthracite culm, Scranton, Pennsylvania. Idem, page 245. *C*. Lykens Valley, Pennsylvania. *D*. Schuylkill coal, Pennsylvania. *E*. Cameron coal, Pennsylvania. Analyses *C*, *D*, and *E* by A. S. McCREATH, Second Geol. Survey Pennsylvania, Vol. MM.

Table XIII[1] shows the ultimate analyses of anthracite coals, ash, sulphur, and moisture being excluded.

TABLE XIII.—ULTIMATE ANALYSES OF ANTHRACITE

	A	*B*	*C*	*D*	*E*	*F*	*G*
Carbon	91.47	92.73	93.90	94.63	94.68	97.46	93.50
Hydrogen	4.25	3.37	3.22	2.73	2.29	0.61	2.81
Nitrogen	1.64	0.85	1.00	1.36	0.76	0.35	0.97
Oxygen	2.64	3.05	1.88	1.28	2.27	1.58	2.72
	100.00	100.00	100.00	100.00	100.00	100.00	100.00

A. Semi-anthracite; the same as *A*, Table XII. *B*. Welsh anthracite, analysis by F. VAUX. *C*. From Scranton, Pennsylvania, Coal B, Table XII. *D*. From Mauch Chunk, Pennsylvania, Analysis by J. PERCY. *E*. From Province of Hunan, China. Analysis by F. HAEUSSERMANN and W. NASCHOLD. *F*. From the Bajewka, Ural. Analysis by ALEXEJEFF. *G*. Average of 16 analyses of anthracite compiled from different sources.

Distribution of American Coals.—With some unimportant exceptions, the anthracite coals are found in five small coal fields located in eastern Pennsylvania, and in the list here given, they are placed in the order of their hardness.

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[1] Data of Geochemistry, 1908, page 656.

1. Lehigh, or Eastern Middle Field.
    - Green Mountain District.
    - Black Creek District.
    - Hazelton District.
    - Beaver Meadow District.
    - Panther Creek District.
2. Mahony, or Western Field.
    - East Mahony District.
    - West Mahony District.
3. Wyoming, or Northern Field.
    - Carbondale District.
    - Scranton District.
    - Pittston District.
    - Wilkesbarre District.
    - Plymouth District.
4. Schuylkill, or Southern Field.
    - East Schuylkill District.
    - West Schuylkill District.
    - Louberry District.
5. Lykens Valley or Southwestern Field.
    - Lykens Valley District.
    - Shamokin District.

Anthracite coal is also found in Wythe and Pulaski Counties, Virginia; along the border of Little Walker Mountain, and in Gunnison County, Colorado; the anthracite districts in Virginia are, however, limited, and in Colorado the quality is found to vary to a considerable degree in neighboring beds, this variation being also found in the same beds.

The semi-anthracites are located in a few small fields in the western portion of the anthracite field, the largest of these beds being the Bernice field in Sullivan County, Pennsylvania.

The semi-bituminous coals are found along the eastern edge of the great Appalachian Field; starting with the counties of Tioga and Bradford in northern Pennsylvania, this bed runs southwest through Lycoming, Clearfield, Center, Huntingdon, Cambria, Somerset and Fulton Counties in Pennsylvania; continuing through Allegheny County, Maryland; Buchannan, Dickinson, Lee, Russell, Scott, Tazewell and Wise Counties in Virginia; Mercer, McDowell, Fayette, Raleigh and Mineral Counties in West Virginia; finally ending in northeastern Tennessee.

The bituminous fields are largest in the Appalachian district, where they begin near the northern boundary of Pennsylvania, going over into the western portion of that state, and extending southwestward through West Virginia, Maryland and Virginia being touched on their Western borders, thence passing on through southeastern Ohio, eastern Kentucky, and central Tennessee, finally ending in western Alabama, thus stretching across a distance of 900 miles between the northern and southern extremities of the field.

Bituminous coal is also found in the Northern Field, in north central Michigan; and in the Eastern Interior Field, which is somewhat further west, and which is second in importance to the Appalachian Field, this Eastern Interior Field covering nearly the entire State of Illinois, with the exception of the upper northern portion, southwestern Indiana, and the western portion of Kentucky.

The Western Bituminous Field is found extending through central and southern Iowa, western Missouri, southwestern Kansas, eastern Oklahoma, and the western central portion of Arkansas, while the Southwestern Field is confined almost entirely to the north central portion of Texas, where two small fields are also located along the Rio Grande river.

The remaining bituminous fields are found scattered through the Rocky Mountain Region, passing from Montana to New Orleans, the principal ones being found in central, southwestern, and southern Montana; northern, northeastern, southern, and southwestern Wyoming; southern, northern, northwestern, western, and north central Colorado; Eastern and southwestern Utah; northern, northwestern, and southern New Mexico, while a few small fields are scattered along the Pacific Coast in western California, southwestern Oregon, western and northwestern Washington.

The greater portion of the coals produced in the latter fields mentioned above lie on the border between bituminous coal and lignite, being usually known as a low-grade bituminous, or black lignite.

The lignites are found in Wyoming, New Mexico, Arizona, Utah, Montana, North Dakota, Nevada, California, Oregon and Washington, and many of the fields given above as containing bituminous coal also contain lignite. Some lignite is also found in portions of eastern Texas and in Oklahoma.

Coal has also been found in Alaska, and these new fields will undoubtedly prove of great value, high-grade coals being found at present in the Bering River Field, near Controllers Bay, and in the Matanuska Field, at the head of Cook's Inlet, both fields containing anthracite as well as high-grade bituminous coals.

CHAPTER II

OXIDATION AND SPONTANEOUS COMBUSTION

Oxidation accompanied by the production of heat may be broadly termed combustion, even though the process of oxidation be a very slow one, thus also producing heat at such a very slow rate that it is not indicated by a rise in temperature, and the theory that the rate, or rapidity of oxidation, of a combustible during the period preceding ignition, is one of the principal factors which determine the ease of ignition, is generally accepted as a fact. There is also no doubt of the fact that the oxidation of coal is not confined to such temperatures as produce active burning accompanied by flame, because coal exposed to the air will undergo oxidation at ordinary temperatures.

This fact has led to the statement that the weathering of coal with its consequent reduction in value as a fuel is not due to a loss of the volatile combustible gases in the coal, but principally to the combination of the coal with oxygen, this combination being formed in a way which causes a large portion of the oxygen to be retained by the coal, thus causing a portion of the combustible constituents to become inert.

Experiments have shown that a coal which is known to have a high coking value loses a great deal of this quality by being subjected to preliminary heating in air for a short period, or by exposing the coal to the weather for a long period, this loss being due to oxidation.

This slow oxidation of the coal, or of its mineral impurities, by the oxygen of the air, produces a spontaneous heating of the coal, this heating not being due to a process of fermentation in the coal itself, as was formerly supposed, it being a well-known fact that spontaneous combustion cannot occur when air is not present.

Coal at ordinary temperatures absorbs oxygen from the air and thus increases its weight, but the nature of this reaction has never been clearly demonstrated, although it has been suggested that certain constituents of the coal unite with oxygen in such manner as to produce humic acid or derivatives closely related to it.[1]

The spontaneous combustion of coal has been widely investigated, but the literature on this subject is scattered and difficult to secure; Messrs. S. W. Parr and F. W. Kressmann, of the University of Illinois, in their researches on this subject gathered a great deal of this early data and published it in the bulletin of the University, Vol. VIII, No. 16, the following statements being quoted from same.

Grundmann's work, published in 1862, was probably the first serious attempt to study the weathering losses of coal. He found that while the coals lost gaseous constituents, they gained in hydroscopic moisture and so the specific gravity of the coal remained constant. He states that large pieces weather less than fine coal, and that rain and increased temperature accelerate the weathering. Using the percentage of ash as a basis to calculate the change in weight that took place during storage, he found that a pile of 300 tons lost 58 per cent. in weight in 9 months.

Reder in 1866 was among the first to follow up Grundmann's work. He found that storing coal for a year caused a deterioration in the heating value; but, contrary to Grundmann's results, he found no appreciable loss in weight.

In 1865, Varrentropp reviewed Grundmann's work of 1862. Varrentropp and other later workers could not duplicate the results of Grundmann, and ascribe his errors to poor sampling. Varrentropp found from his experiments in storing coal that the oxidation of fresh coal is very rapid, and within the heap leads to a rise in temperature, which, however, is very far from the ignition point. He states, however, that temperatures reaching the ignition point and causing combustion have at times been found.

L. Thompson in 1865 found that dry coal in storage for 6 months lost one-thirteenth of its calorific value; if, however, the coal was moist, the loss was much greater. He divides the weathering of bituminous coals into two classes known as "dry rot" and "wet rot." The latter he considers to be a sort of fermentation in which a part of the hydrogen escapes and some of the carbon is oxidized to carbon dioxide; in this way the coal loses calorific value in two directions.

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[1] Bureau of Mines, Tech. paper No. 65. Study of the Oxidation of Coal, H. C. Porter and O. C. Ralston.

FLECK in 1865 examined a series of six Saxon coals which had been preserved in a cabinet since 1856. The results of his examination were compared with analyses made 9 years previously, when, it is presumed, the coals were in a fresh state. After such an exposure in a dry place, three of the number, it was inferred, showed an important increase in proportion of ashes together with a corresponding decrease of organic matter, while the ashes of the remaining three appeared to have decreased. The explanation of such unlike results was sought by FLECK in the lack of uniformity of the related samples, especially as to the ratio of ashes.

An increase of oxygen and of indisposable hydrogen was observed in each case, together with a loss of carbon and of disposable hydrogen. Hence the conclusion that, at ordinary temperatures, bituminous coal sustains a loss in carbon and of disposable hydrogen, and proportionately a loss in calorific value, which is increased in proportion to the addition of combined water.

The most careful of the early experiments made on this subject were those of DR. RICHTER from 1868 to 1870. He concluded that the weathering of coal was due to the absorption of oxygen, part of which goes to the oxidation of carbon and of hydrogen in the coal, and part of which is taken into the composition of the coal itself; that if the heap becomes warm, either through this process or any other cause, the action is accelerated, but then falls off, and becomes so slow that the changes effected within a year become difficult to estimate; that moisture, as such, has not direct influence upon the process apart from the presence of pyrites or from the coal crumbling down more rapidly when wet than when dry and, therefore, more rapidly heating up; that if the temperature does not go beyond certain limits (338° to 374°F., 170° to 190°C.) there is no appreciable loss of weight but rather, on the whole, a gain through the absorption of oxygen. Also, that it is not necessary to explain the loss in heating value, of coking power, and of gas yield by any theory of new arrangements of atoms, for it is sufficient to look at the absolute loss of carbon and hydrogen and the increase in the proportion of oxygen. At a later date, he concluded that large coal was affected less than small, not because it had less surface, but because small coal was a more active absorbent of oxygen, and, therefore, became more rapidly heated; that air-ways in a heap would have to be very numerous in order to prevent any rise in temperature; and that the heap ought to be so constructed as not to allow air to get from the airways into the bulk of the coal.

Haedicke in 1880 assigned to pyrites the leading part in spontaneous ignition, conditioned on allowing the temperature to rise sufficiently.

Prof. Fischer of Göttingen, in his work on this subject prior to 1901, concluded that storage depreciation and spontaneous ignition were phenomena of oxidation; the part which is played by iron sulphide was disputed, but the variances which have given rise to the uncertainty was due to the differences between the different sulphides of iron present in coal. He claims that marcasite, for example, is much more weatherable than ordinary pyrite. Actual wetting is much more promotive of oxidation of the iron sulphide than heating in dry or even moist air. Besides this, **Fischer** finds that many coals contain sulphur in the form of unsaturated organic compounds. He finds that those coals which rapidly absorb bromine are those which are most liable to rapid oxidation and spontaneous ignition; as a practical test, he recommends shaking a gram of the finely powdered coal with 20 cc. of a half normal solution of bromide for 5 minutes. Then, if the smell of bromine has not disappeared the coal may safely be put in storage; if it has, the coal should be used as soon as possible. The absorption of oxygen by the unsaturated organic compounds referred to is accompanied by a gain in weight and at the same time by a rise in temperature; and the warmer the mass the more rapidly does the oxidation go on, so that the temperature is apt to rise to a dangerous extent. Concurrently with the oxidation of these products, is the oxidation of another set of organic compounds with evolution of carbon dioxide and water vapor, and this results, not in a gain, but in a loss of weight. Whether the mass as a whole gains or loses weight is, therefore, a question of the composition of the coal, but the loss in weight is in no case great. Covering wet slack coal with other coal is apt to produce spontaneous ignition; the danger here appears to arise from the sulphide of iron rather than from the organic compounds. **Fischer** regards ventilation of the coal heap with suspicion, not because the idea in itself is a wrong one, but because it is not practicable to ventilate the whole heap efficiently. He claims that the coal should be stored dry and kept dry under cover, and in layers not too deep.

In a paper read before the German Gas Association in 1900, Herr **Sohren**, Manager at Bonn, stated that it is no longer possible for many reasons to work gasworks with a supply of coal renewed from month to month, and that all questions affecting storage have, therefore, a continually increasing importance. Undoubtedly there is a greater or less depreciation in the quality of coal kept in store, and the causes of this have attracted a good deal of attention, though, on the whole, it is surprising to find to how great an extent the study of the chemistry of coal has been neglected. Questions of this nature assumed importance in connection with the spontaneous ignition of cargoes of coal in ships; in 1874, out of a list of 4485 coal laden ships, no fewer than 60 caught fire.

In 1906, **Prof. Lewes** of the Royal Naval College, England, stated that, in the spontaneous ignition of coal stored in bulk, the oxidation of the pyrites present plays only a very subsidiary part, the chief factor being the surface condensation of the oxygen in the pores of the coal and the action of the condensed gas in effecting the oxidation of the hydrocarbons present in the coal. While sufficient ventilation to prevent any considerable increase in temperature within the mass is effective in preventing ignition, it is practically impossible to maintain this condition in large coal stores, or in a cargo on board ship, where usually only enough air enters to lead to dangerous heating.

A. O. Doane in 1904 stated that the amount of moisture present in a bituminous fuel after drying in the air is a measure of the risk of spontaneous ignition when the fuel is stored; bituminous coals containing over 4.75 per cent. of moisture are dangerous. Coal bins should be of iron or steel, protected by concrete, and should be roofed over. Free air passages should be provided around the walls and beneath the bins so as to keep them cool, and the depth of coal should not be over 12 ft. The customary method of providing air passages in the body of the fuel is useless, since it tends merely to accelerate oxidation and does not produce a sufficient current of air to keep the temperature down. Cracks or joints in the walls of the fuel bin increase the risk of spontaneous ignition for similar reasons.

Durand has explained the spontaneous ignition of coal as being due to the presence of pyrites, which on oxidation under suitable conditions inflames, and then sets fire to the coal in which it is imbedded.

According to FAYOL's experiments, however, the real cause of spontaneous combustion is the oxidation of the coal itself and not that of the pyrites. The absorption of oxygen by coal takes place more or less readily, depending upon the initial temperature and upon the coal being more or less finely divided. Lignite in the shape of fine dust takes fire at 150°C. (302°F.), gas coal at 200°C. (392°F.), coke at 250°C. (482°F.), and anthracite at 300°C. (572°F.) or above. On heating a mixture of finely powdered coal and pyrites up to 200°C. (392°F.) for 4 days, the coal took up 6 per cent. of oxygen, while the pyrites absorbed only 3.5 per cent. Hence coal absorbs oxygen much more energetically than pyrite does, a fact which was confirmed by the following experiment: 900 grams of powdered coal and 3350 grams of powdered pyrites were placed in tin cans and put in a drying chamber. Up to 135°C. (275°F.) both behaved similarly, but from there on the temperature of the pyrites remained almost stationary while that of the coal quickly rose, ignition taking place a few hours afterward. Two other samples of coal and pyrites were placed in a chamber at 200°C. (392°F.). The temperature of the coal quickly increased. After 40 minutes it reached 200°C., and the coal took fire, while the pyrites had at the same time been raised only to 150°C. (302°F.). The ignition of the coal was not at all hastened by an admixture of pyrites.

FAYOL, as a result of some of his former experiments, stated that "the influence of wet weather on heaps of coal has not been sufficiently marked to be observable." On the other hand, much evidence has been given which is absolutely opposed to the above statement. In a letter to the English Commission in 1876, MR. POOLE, Inspector of Mines, Nova Scotia, makes the following remark:

> "Heat has been observed in piles of slack coal that have been accumulated during the summer for the use of engines during the winter, when the season has been showery, but not when it has been dry."

Much similar evidence was offered the English Commission, for, out of the twenty-six answers to questions as to the effect of moisture, every reply was to the effect that moisture was a source of danger. Similar evidence was given at the inquiry held by the New South Wales Commission, although, on cross examination, it was brought out that this view was more a matter of impression than of definite knowledge.

FAYOL also found that the coal of Commentry never took fire by exposure, when the heaps were less than 6 ft. 6 in. deep, no matter what the lateral dimensions of the heap might be. He also found that a mixture of dust and fragments was the most dangerous.

MR. BUSH, engineer of the Australian Gas Company, states that he maintains two coal stores, one having a coal depth of 14 ft., and the other of 20 ft.; the latter pile requires a great deal of attention to prevent the coal from firing, while in the case of the former no instance of firing from spontaneous ignition has occurred all conditions, except depth of coal, being the same in both piles, and he attributes the behavior of the coal in the two piles to the difference in the depth of the coal.

F. M. GRISWOLD of the Home Insurance Company, in 1904, stated that spontaneous ignition is more marked in free-burning or so-called "high steaming coals" including "gas coals." These coals usually contain a large amount of volatile matter and small amounts of oxygen. The tendency toward spontaneous ignition increases when sulphur or lignites are found in the coal, and especially when pyrite is present. Slack, mine run, and dirty coal are dangerous because of the fine particles present, the latter gradually sifting to the bottom and being compressed. According to MR. GRISWOLD, a satisfactory explanation of spontaneous ignition has as yet not been made. The best authorities say it is due to chemical changes in the substance of the coal, resulting from the absorptive powers of carbon, the latter increasing with the rise in temperature. Rise in temperature may be due to the chemical action caused by slow oxidation or to the mechanical force of pressure; both of these conditions may be stimulated by moisture and the presence of pyrites. Some claim that over 2.5 per cent. of sulphur in the form of pyrite is dangerous. Various tests have been proposed to determine the liability of a coal to heat, such as the gain in weight of the coal at 250°F. (121°C.) and the absorption of bromine, but these tests are not valuable, as it is difficult to tell how much oxygen the coal has already absorbed. He recommends that no wood be used in the construction of bins, that all iron work be covered with concrete, that no steam pipes or flues be near the bins, that a good circulation of air be about the sides and under the bins, and that the coal should not be piled to a depth greater than 12 ft. He also recommends that the bins be roofed, and that they contain pipes in which thermostats should be placed, and when the latter register 140°F. (60°C.), something should be done to stop the rise in temperature.

GROVES and THORP state that gases occluded in the crevices or cavities of the coal escape during mining and continue to do so after storage. The inflammable nature of these gases makes them a source of great danger, for it is believed that they are the causes of many of the disastrous explosions on board vessels carrying coal as a cargo.

PARR and BARKER, working on the gases occluded in Illinois coal, found that two active processes are set up immediately upon liberation of the coal from the vein. The first is an exudation of hydrocarbons, consisting mainly of marsh gas (CH_4), and the second is an absorption of oxygen. They found that after a period of 2 months practically all of the marsh gas had been liberated and, although the avidity of the fresh coal for oxygen was very pronounced, after a period of 2 months, active absorption of oxygen still took place, the work of R. T. CHAMBERLIN on explosive mine gases and dust confirming the work of these investigators.

Other investigations made by DEUNSTEDT and BÜNZ in 1908, by GRAEFE, ERDMANN, STOLZENBERG, BOUDOUARD and others might be cited, but a summary of the opinions of all of these will clearly show the results of their work; the opinions of many differ as to the real causes of spontaneous combustion, but the leading factors may be summed up as follows:[1]

1. The kind of coal, in regard to its volatile matter.
2. The purity of the coal.
3. The presence of pyrite or other sulphur compounds.
4. The temperature of the coal.
5. The size of the coal.
6. The presence of occluded gases in the coal.
7. The presence of moisture.
8. The accessibility of oxygen.
9. Pressure on the coal.

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[1] Bulletin No. 46, University of Illinois. "The Spontaneous Combustion of Coal," PARR and KRESSMANN.

1. Kind of Coal.—From the ignition temperatures given by Fayol, it may be seen that only those coals (such as lignites, bituminous and semi-bituminous) containing large amounts of volatile matter, are liable to ignite spontaneously, and that anthracite, with its very low percentage of volatile matter, is practically entirely excluded. The results of the work of O. Boudouard, on the coking power of coals, also substantiates this view, for in his work he found that those coals richest in volatile matter (carbohydrates, cellulose, etc.), were most liable to spontaneous combustion.

2. Purity of Coal.—The work of our own government seems to indicate that coals of exceptional purity are more apt to heat up than coals containing large amounts of extraneous matter. This is probably due to the fact that very pure coals are able to condense and absorb the oxygen of the air much faster than other coals and so cause an increase in temperature, which finally results in the chemical combination of the oxygen and hydrogen occluded in the coal. This view is confirmed by the later work of Deunstedt and Bünz, who found that those coals which oxidized and increased in temperature most rapidly were remarkably free from mineral matter and pyrite.

3. Presence of Pyrite.—As to what part sulphur compounds, especially pyrite, play in the spontaneous ignition of coal, opinions differ greatly. Some believe pyrite to be the leading factor, while others believe it plays no part at all, or, if so, ascribe to it a position of minor importance and believe its action to be merely a subsidiary one. The oxidizing action of the air upon pyrites is, however, admitted, and the notion appears to be fairly general and well established that pyritic oxidation tends to raise the temperature of the coal. On the other hand, it is seen from the work of Fayol, Deunstedt and Bünz, Threlfall and others that coals containing pyrite in a quantity too insignificant to be noticed are very apt to ignite spontaneously. The Newcastle coal of New South Wales is also a very good example of this class of coals. Others, however, believe that the only influence of the pyrite is a mechanical one, in which the oxidation of the thin films of pyrite in the coal serves merely to break up the coal.

4. Temperature of the Coal.—Most of the authors agree that the temperature of the coal undoubtedly is one of the main factors in the whole subject of spontaneous combustion, for cases of spontaneous combustion have occurred time and again where probably they never would have occurred if there had not been an initial heating in some way or other. The New South Wales Commission thinks that the initial temperature of the coal at the time of storage or loading is one of the great factors in the subject of spontaneous combustion, the true danger of which has not heretofore been fully appreciated. Sources of heat seemingly insignificant are frequently the cause of bringing the temperature of the coal up to the danger point and so causing coals to ignite which, under carefully regulated conditions of storage, would not be at all dangerous. Numerous cases are on record where the cause of the spontaneous ignition of large coal stores was finally traced back to initial heating from flues, steam pipes, or direct exposure to the sun. This increased temperature, whether coming from outside sources or from physical or chemical reactions within the coal, tends to accelerate the absorption of oxygen and thereby to raise the temperature of the coal. It also tends to drive out the gases occluded in the coal, and so greatly to accentuate the danger of spontaneous ignition.

5. The Size of the Coal.—That fine coal is a more active absorbent of oxygen and more liable to ignite spontaneously than large coal was shown as far back as 1868 by RICHTER. Practically all of the later experiments in this field concede this to be true, for, having a greater surface, the fine particles can absorb much more oxygen than large lumps; and since this rapidity of absorption causes an increase in temperature, which in turn produces favorable conditions for further absorption and for chemical action between the oxygen of the air and the hydrogen of the coal, the danger of the spontaneous combustion is greatly increased.

Throughout the literature, warnings are found cautioning coal users and shippers against storing fine coal. Some have even advocated that only large lumps be transported on shipboard and that these should be carefully packed by hand. The covering of coal piles by slack is to be avoided, also the coal should be handled as little as possible to avoid disintegration, for, in handling, the finer particles gradually sift to the bottom of the pile where they accumulate and are a source of danger inasmuch as at the center and bottom of the pile the most favorable conditions for spontaneous ignition exist. Here the coal is sufficiently insulated by the surrounding coal to retain all the heat that is generated. Air is accessible in sufficient amounts to cause oxidation, and still is not present in large enough amounts to conduct away the heat. The fact that friable coals are more prone to ignite spontaneously than coals not so friable also seems to substantiate the above opinion. DEUNSTEDT and BÜNZ in their work on this subject even went so far as to make a friability test on all the coals they worked with for they conceded the danger arising from the fine coal with its great avidity for oxygen, and recommended that they considered safe to transport on board ship only those coals passing a certain standard in their friability test.

6. Occluded Gases in the Coal.—While it is now a well-known fact that gases of an inflammable nature are occluded in coal, their relation to the spontaneous ignition of coal has not yet been clearly established. Whether the gases occluded in the coal are the real cause of spontaneous ignition is doubtful, but if the coal becomes heated up by oxidation or some other cause to a temperature high enough for the oxygen of the air to unite with these gases, then it is seen that the presence of these gases constitutes a source of danger, and then coals with large amounts of gases occluded in them would be more liable to ignite than coals containing smaller amounts of these gases.

7. The Presence of Moisture.—Opinions differ greatly as to what part moisture in the coal plays in the spontaneous combustion of the same. Some, as DOANE, believe the moisture content of the air-dry coal to be a direct index of its power to ignite spontaneously. Evidence given to the New South Wales Commission and also to the British Commission shows that coal piles are more apt to take fire during warm weather following showers than at other times, although FAYOL, as a result of experimental work, claims that the influence of the weather on coal heaps has not been sufficiently marked to be observable.

ERDMANN and STOLZENBERG believe the spontaneous combustion of coal to be due to the formation of ozone by the action of the sun on warm, sunny days following a rain, when the surface evaporation is especially great. If this is true, moisture employs a more important part in the phenomena of spontaneous combustion than has heretofore been ascribed to it.

That the presence of moisture materially assists the pyritic oxidation is generally conceded, although whether it causes an increase in temperature or merely a disintegration of the coal due to the formation of ferrous sulphate is a matter of dispute. Others believe that the only part that moisture plays is a mechanical one, where alternate freezing and thawing break up the coal into smaller particles and so expose more surface to the oxygen of the air. Again, it is thought that, aside from increasing the pyritic oxidation, moisture acts as a catalytic agent between the carbon and the hydrogen or unsaturated bodies of the coal and the oxygen of the air. Perhaps, as **ERDMANN** and **STOLZENBERG** have suggested, this reaction may consist of the formation of ozone which is immediately absorbed by the coal. This idea of catalytic action on the part of the moisture is substantiated to some extent by the fact that some coals containing minimum amounts of pyrite are nevertheless very liable to spontaneous ignition; and coals of this class, viz., the Newcastle and Scotch coals, have been known to cause fires.

8. The Accessibility of Oxygen.—That the combination of oxygen with the constituents of the coal causes a rise in temperature seems to be firmly established. Which particular constituent is the cause of the rise in temperature has not, however, been shown with any great degree of certainty. The presence of humic acid in the oxidized coal leads one to believe that the oxygen combines with some of the unsaturated humus bodies, such as polysaccharides. That this humus substance is an unsaturated body is shown by the fact that it absorbs large amounts of bromine without the evolution of hydrobromic acid. In fact, **FISCHER** goes so far as to devise a practical test to determine the safety of a coal by means of this reaction with bromine.

The idea is also held that the oxygen of the air combines directly with the carbon and hydrogen of the coal and so causes an increase in temperature. If amorphous carbon (charcoal and lampblack) can be oxidized to carbon dioxide on exposure to the air by means of bacteria, as has been proved by **POTTER**,[1] the oxidation of the carbon of the coal is very probable. The presence of oxygen is therefore to be avoided and the old idea of thoroughly ventilating the coal piles by free access of air is gradually being dropped and, at the present, authorities deem it advisable to keep the coal away from the air as much as possible, either by submerging it under water or storing the coal in covered concrete bins. If ventilation is used to lower the temperature of the coal, it should be through pipes, so that the cooling air cannot come in contact with the coal at all.

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[1] Proc. Royal Soc., London (B), Vol. 80, page 196.

9. Pressure on the Coal.—The belief that pressure on the coal is one of the leading factors in the spontaneous ignition of coal seems to be gaining ground, and because of this fact it is advocated that coal heaps should not be any higher than 15 to 20 ft. The experience of MR. BUSH (Australia) in this respect is certainly very startling, and in a store of that size the question is very probably more one of pressure than of radiation, although in smaller storage units it would probably be hard to decide whether the cause of the spontaneous combustion was due to pressure or lack of proper insulation of the heat generated within the pile.

PARR and KRESSMANN in their further studies on this subject made quite a number of experiments at the University of Illinois,[1] and their conclusions are quoted below.

PARR AND KRESSMANN'S CONCLUSIONS

1. The oxidation of coal is continuous over a wide range of time and conditions, and begins with the freshly mined coal at ordinary temperatures. A number of oxidation processes are involved, which are more or less distinct in character, some being relatively slow and moderate in form, while others are rapid and vigorous in their action.

2. In general it may be said that for a given coal a point exists, as indicated by the temperature, below which oxidation is not ultimately destructive. The continuance of this point is dependent upon certain accessory conditions; if these conditions are withdrawn, the oxidation ceases. On the other hand, above this critical point, which is best indicated by temperature, oxidation is ultimately destructive and is characterized by the fact that it does not depend for its continuance upon external conditions, but is self-propelling, or autogenous.

3. The point of autogenous oxidation, while varying for different conditions, may be indicated by temperatures of the mass ranging from 140° to 160°C. (284° to 320°F.) in an atmosphere of oxygen, or approximately between 200° to 275°C. (392° to 527°F.) in oxygen diluted with nitrogen, as in air, depending

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[1] Bulletin No. 46, University of Illinois. "The Spontaneous Combustion of Coal."

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¹ Proc. Roya

er. very largely dependent upon the freshness of the coal and upon the fineness of division (see *b* above), and should be looked upon as a contributing factor, though in coals of the Illinois type, at least, with their high per cent. of sulphur, this action should doubtless be considered second in importance to that of iron pyrites.

(*d*) *Iron Pyrites.*—The presence of sulphur in the form of iron pyrites is a positive source of heat due to the reaction between sulphur and oxygen. This may be conveniently referred to as the second stage in the process of oxidation. Here again rapidity of oxidation is directly dependent upon fineness of division. Since coals, as a rule, have a much higher earthy or ash content in the fine duff, and since iron pyrites is a large component of this material, it follows that the presence of dust or duff in all coals of the Illinois type is a positive source of danger. Since coals of the Illinois or mid-continental field have in the larger number of cases iron pyrites averaging over 6 per cent., or, as sulphur above 3 per cent., the heat increment from the oxidation of only one-fifth of this material is sufficient to raise the temperature of the mass approximately 70°C., assuming that there is no loss by radiation. Under usual conditions, and especially considering the greatly accelerated rate of chemical activity accompanying a rise of temperature, this oxidation may proceed with such rapidity that the heating up of the mass will be but little affected by loss of heat due to radiation, except in relatively shallow piles.

(*e*) *Moisture.*—Moisture, while essential to pyritic oxidation, is given separate mention because its importance is apt to be under-estimated.

Any coal with conditions favorable to oxidation will be facilitated in that action by moisture. It is to be noted in this connection that the normal water content or vein moisture of coals in this region (Illinois) is rarely below 10 per cent., and ranges usually from 12 per cent. to 15 per cent. The presence of such water must be borne in mind in considering the likelihood of chemical activity on the part of the pyrites present. Without exception, in all the series of tests, *the wetting of the coal increased the activity as shown by the ultimate temperature.*

(*f*) *The Oxidation of Carbon and Hydrogen.*—A third stage of oxidation of the carbonaceous material exists by reason of the tendency of certain of the hydrocarbon compounds of coal to oxidize with the formation of CO_2 and H_2O at temperatures in excess of 120° to 140°C. (248° to 284°F.). Though this type of oxidation does not take place appreciably at ordinary temperatures, it must be looked upon as an exceedingly dangerous stage in the process of oxidation, owing to the very much higher quantity of heat which is discharged by the oxidation of carbon and hydrogen; so that the temperature of autogenous action, although ordinarily occurring at a higher point by 100°C. (212°F.) or more, may be quickly attained as a result of this form of oxidation. Any initial heat increments, therefore, which threaten to bring the chemical activities along to the point where the oxidation processes invade the carbonaceous material in this manner must be looked upon as dangerous. For example, any of the initial or contributory processes which result in raising the temperature of the mass 50°C. (122°F.) above the ordinary temperature, would, in all probability, have enough material of the sort involved in such action to continue the activity until another 50° had been added, which would thereby attain to the condition wherein this third type of oxidation would begin.

(*g*) The fourth stage of oxidation may be indicated as occurring at temperatures above 200° to 275°C. (392° to 527°F.) and differs from the previous stages in that the action is autogenous and not dependent upon other sources of heat to keep up the reacting temperature. Activity in this stage is further accelerated by the fact that above 300°C. (572°F.) the decomposition of the coal begins, which is exothermic in character, thereby contributing somewhat to a further increase in temperature. The ignition temperature is reached at a point still further along, usually in excess of 300° to 400°C. (572° to 752°F.).

5. The above formulation of the various stages and types of oxidation clearly indicates the principles which must be observed in any attempt at the prevention of spontaneous combustion. The following enumeration, therefore, of preventive or precautionary measures is to be considered as suggestive rather than complete in character.

(*a*) The avoidance of an external source of heat which may in any way contribute toward increasing the temperature of the mass is a first and prime essential.

(*b*) There must be an elimination of coal dust or finely divided material. This will reduce to a minimum the initial oxidation processes of both the carbonaceous matter and the iron pyrites. These lower forms of oxidation are to be looked upon as forces, without which it would be impossible for the more active and destructive activities to become operative.

(*c*) Dryness in storage and a continuation of the dry state, together with an absence of fine material, would practically eliminate the oxidation of the iron pyrites.

(*d*) Artificial treatment with specific chemicals or solutions intended to act as deterrents does not offer great encouragement, though some results seem to warrant further trial in this direction.

(*e*) By means of a preliminary heating, the low or initial stages of oxidation are effected. These sources of contributory heat being removed, the forms of destructive oxidation are without the essential of a high starting temperature and are therefore inoperative. Whether such preliminary treatment is within the realm of practical or industrial possibility could not, of course, be determined within the scope of these experiments.

(*f*) The submerging of coal, it is very evident, will eliminate all of the elements which contribute toward the initial temperatures. As to its industrial practicability, like the conditions under *e* above, it can best be determined by actual experience.

(*g*) Other processes may be suggested by the formulation of the principles involved. Such, for example, would be the distribution throughout the coal of cooling pipes through which a liquid would circulate having a lower temperature than the mass. This would serve to carry away any accumulation of heat and confine the oxidation to the lower stages only. On the contrary, the proposition sometimes made to provide circulating passages for the transmission of air currents is of questionable value, since it may result in the contribution of more heat by the added accessibility of oxygen than will be carried away by the movement of the air.

T. F. WINMILL[1] states that the rate of oxidation of coal dust maintained at a temperature of 30°C. (86°F.) was investigated in the laboratory, different coals from the Barnsley seam being used. Oxidation of fresh coal took place during the first few hours, the amount absorbed being stated as proportional to the amount and character of the carbonaceous matter in the coal. This rapid reaction ceased in 48 hours, when 0.1 cc. of oxygen was absorbed by 100 grams of coal per hour, but it was found that under certain conditions the temperature of the coal might

[1] Jnl. Soc. Chem. Ind., 1913, page 970.

be raised by this rapid initial oxidation to such an extent as to increase the velocity of the slower normal reaction to a dangerous degree, and that this initial oxidation could not be stopped by merely reducing the percentage of oxygen in the air. When coarser dust having only 1/400 of the former surface was used, the rate of oxidation was reduced only by 28 per cent., but with increasing temperature the rate of oxidation increased rapidly.

O. Binder[1] states that the old view as to the spontaneous ignition of coal being due to the oxidation of the pyrites is no longer tenable, claiming that the coals most subject to spontaneous combustion are not always those which contain the most sulphur, and even the instantaneous combustion of 1 per cent. of sulphur would raise the temperature of the coal by only about 135°F. (57°C.). He further states that the combustion of the sulphur must be spread over quite a period, thus precluding the possibility of this fact setting up spontaneous combustion. Observation shows that spontaneous combustion occurs only in the interior of the coal pile, and then only in small coals where the access of air is restricted. As regards the theory that spontaneous ignition is due to the absorption of oxygen by the coal, he claims that laboratory experiments have shown that coal may absorb as much as 4 per cent. of oxygen under favorable conditions. He claims that this theory was due to experience with very finely divided fresh wood characoal, the structure and porosity of which differs greatly from that of coal. Finally, he states that spontaneous ignition may be regarded as being due to continuous decomposition and change in the coal, these phenomena being accelerated by releasing the coal from the pressure to which it was subjected from the material overlying the coal seam. The gas resulting from the first stages of this decomposition of the coal is marsh gas, and this may easily form an explosive gas mixture above the coal pile. If the pile is ventilated, it will prevent the accumulation of large quantities of marsh gas above the coal, but on the other hand ventilation may only quicken the heating of the coal, thus setting up spontaneous combustion, the latter resulting in the production of larger volumes of marsh gas. In order to avoid spontaneous combustion in coal bins, etc., Binder proposes supplying the bins with a system of pipes through which cold brine is circulated, thus preventing an increase to any considerable extent of the temperature of the stored coal.

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[1] London Jnl. of Gas Ltg., 1912, page 750.

BINDER's proposition to use brine in pipes to cool the coal pile is seconded by S. F. WALKER who states that after a series of investigations covering a long course of years, he has concluded that the presence of iron pyrites in the coal is not the principal source of heating the coal, but that this heating is due to the absorption of oxygen followed by oxidation of the carbon as combustion, this process being slow at first, but increasing in geometric ratio with the temperature. He therefore claims that the conditions governing spontaneous combustion are a certain initial temperature, which may be produced by any means whatever, and the absorption of oxygen. Fine coal, which exposes a larger suface to the air than does coarse coal, tends to heat more quickly, and as, due to its nature, it is more closely packed, it retains the heat generated. The creation of a pressure upon the lower layers of the coal in the pile by storing to a considerable height leads to two of the conditions governing spontaneous ignition, viz., a higher temperature and the formation of dust.

WALKER claims that if this reasoning is correct, spontaneous ignition can be avoided by preventing a temperature rise in the coal pile, suggesting the use of a refrigerating plant for this purpose. He suggests that the coal be stored over a series of refrigerating pipes, or between vertical hollow plates, through which the cooling liquid is circulated, sending the cooling fluid through the system only when the temperature of the coal pile requires it.

HORACE C. PORTER[1] gives some very interesting data on this subject, gathered from coal consumers who store as high as 500,000 tons in one pile. The principal object of the investigations recorded by PORTER, was to determine whether the loss in heat units in coal stored during a certain period of time meant a loss of money to an amount sufficient to justify the cost of under-water storage, or some other protective equipment. The practice of storing coal under water has not become general in the United States, but a few instances of commercial trials are published. In 1907 the Western Electric Company installed a submerged storage plant at Hawthorne, Ill., the plant being of concrete construction and having a capacity of 10,000 tons. After 2 years it was found that the submerged coal had a heating value of about 2 per cent. more than that of a similar coal which had weathered

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[1] Electrical World, 1912, page 934.

2 years. PORTER states that this difference hardly justified the $70,000 expenditure for the construction of the plant plus 7 cts. per ton for hauling, but that the insurance against all risk of spontaneous ignition in the stock might be considered to have been sufficient justification in itself. The Bureau of Mines recently concluded a number of tests which extended over a period of 2 years to determine the amount of deterioration caused by weathering four different kinds of coal, the coal used for the tests being New River coal from the Sun mine, Fayette County, W. Va., known as a smokeless coal, containing about 20 per cent. volatile matter; Pocahontas coal, a coal similar to New River; Pittsburgh gas coal, which was examined in order to study its deterioration in gas-producing qualities, and Sheridan (Wyo.), sub-bituminous coal, or black lignite. The loss found in the New River coal after 2 years of storage varied between 1.8 per cent. and 0.9 per cent. for weathered coal, while that stored under water showed practically no loss. Pocahontas coal lost 0.4 per cent. and Pittsburgh gas coal lost practically nothing in 1 year's weathering. The Wyoming coal showed a loss in one case of 2.5 per cent. in the first 3 months and over 5 per cent. in weathering for 2 years and 9 months. This coal slacked badly on the surface, but this slacking did not penetrate more than 18 in. There was no apparent slacking of lumps or physical deterioration in the run of mine with the New River and Pocahontas coal, but in all cases a ¼-in. crushed coal exposed to the weather deteriorated in heat value more than did the run of mine, this deterioration being more marked in warm climates. Reports of other tests seem to corroborate in some degree the results mentioned above. At Sitka, Alaska, the U. S. Navy Department kept Pocahontas coal under cover for 4 years, at which time this coal showed a loss of 0.7 per cent., while the coal recovered from the wreck of the battleship Maine in Havana harbor, and which had been submerged for 14 years, showed a loss of only 1.9 per cent. The origin of this latter coal has never been established, but it is presumed that it was coal from the New River district, West Virginia.

According to PORTER, spontaneous combustion results from an accumulation in a combustible material of the heat produced by oxidation, or the heat of other chemical reactions, of fermentation, or of bacterial action. Tests have clearly shown that coal exposed to air is subjected to oxidation, and that some coals are affected more rapidly than others in this particular, but some coals which during these tests showed little or no heat development proved in practice dangerous to store, which condition led to a consideration of factors other than those of oxidation, these factors being the effect of the physical conditions of storage on accumulation or dissipation of heat. A report made by a committee of the Railway Maintenance of Way Association stated that in the committee's opinion the presence of dust in a coal pile was the prime cause of spontaneous combustion, this statement being corroborated by other authorities. A very instructive instance from commercial practice is presented by the Calumet and Hecla Mining Company, near Haughton, Mich.; a stock of 100,000 tons of Pittsburgh run-of-mine coal was stored at the Tamarack coal dock, spontaneous fires giving continuous trouble in this stock until some time ago, when it was decided to screen the coal before storing, only the lumps being stored, the fine coal being used immediately. Practically no trouble with fires has been encountered since this system was adopted.

Porter states that the cause of an open coal pile becoming heated spontaneously after a heavy rain or alternate wetting and drying, is possibly due to the washing down of dust into a compact layer from 2 to 3 ft. below the surface where its lower portions start to heat and where they are insulated from cooling air currents. He also states that some coals are more liable to heat, or to undergo spontaneous ignition, than others, and that storage piles having a depth of more than 15 ft. are dangerous; also that a mixture of fine coal with a small quantity of lump coal affords a greater opportunity for heating, as this mixture allows access for a limited quantity of air. In concluding, Porter states that freshly mined or freshly crushed coal heats easily, but that if it is rehandled after a 2 months' storage, risk of spontaneous ignition can probably be avoided.

CHAPTER III

COKING AND GAS COALS

When coal is carbonized, no matter whether the process is carried out in the conventional retort of a gas works, or in a by-product coke oven, we find that the resultant distillation is a very complex process, as it is necessary to distill the volatile products at temperatures varying from 250 to 2200°F. (121° to 1200°C.). This distillation leads to the production of volatile products, and the composition of these products is dependent not only upon the temperature within the substance being distilled, but even to a greater extent upon the temperature of the region through which these products are compelled to pass, because the ultimate composition of the resultant is a product due to the distillation of many coal substances, each of these substances probably distilling at a different temperature, as well as being a product of the time and conditions of temperature to which the first products were subjected after being driven from their original state, or condition, in the coal.

The time and temperature conditions, to which the substance to be distilled is subjected, are determined by the bulk of the mass to be heated and by the rate at which heat is supplied from without, and a study of carbonization must necessarily presuppose a knowledge of the various coals to be carbonized, especially as this relates to their volatile products resultant at various temperatures, this knowledge leading to data which readily influence working conditions.

When coal is distilled, or when it is heated without the access of air, we find that carbon dioxide is the first gas liberated, and this is given off when the coal is only gently heated, being followed by saturated paraffin hydrocarbons, these latter constituents being freely produced by the distillation of gas and coking coals. The further heating of the hydrocarbons at high temperatures causes their decomposition, and due to this decomposition we secure the well-known consituents, coal gas and coal tar.

The principal diluents, or anti-calorific substances in coal are oxygen, nitrogen, ash, and moisture, and besides being a diluent, the presence of oxygen also renders ineffective a certain amount of carbon, hydrogen, and sulphur, thus reducing the heating value by rendering a certain portion of these fuel constituents inert. When coal is distilled, the volatile constituents are first driven off, and due to the fact that hydrogen is of a higher calorific standard than an equivalent weight of carbon, we find that the distribution of the oxygen between the carbon and the hydrogen in the volatile products has an important and direct bearing on the calorific value of the fuel, and available hydrogen has a greater value than available carbon.

In DULONG's method of calculating the heat value of a fuel, the available hydrogen is reduced by an amount equivalent to all of the oxygen, but many fuels show a different result in practice, probably due to the fact that these coals distribute a large proportion of the oxygen content to carbon instead of to hydrogen.

H. C. PORTER and F. K. OVITZ[1] experimented with a number of low grade, but highly oxygenated coals, and their results showed that nearly two-thirds of the oxygen appeared in the

TABLE XIV.—OXYGEN RELATIONS IN VOLATILE MATTER
(Values are percentages of air-dried coal)

Coal	Per cent. of oxygen compounds in volatile matter			Oxygen in CO and CO_2	Oxygen in water	Total oxygen in dry coal
	CO_2	CO	Water of constitution			
No. 16 (Pocahontas):						
400-gram tests	0.44	1.21	0.1	1.01	0.09	3.18
10-gram tests	0.90	1.74	1.5	1.65	1.33	3.18
No. 3 (Connellsville):						
400-gram tests	0.72	......	3.7	....	3.29	5.23
10-gram tests	1.04	2.33	3.5	2.08	3.10	5.23
No. 1 (Zeigler, Ill.):						
10-gram tests	1.66	4.90	6.7	4.01	5.95	9.12
No. 11 (Sheridan, Wyo. air-dried):						
400-gram tests	8.60	6.90	7.5	10.19	6.67	16.63
10-gram tests	8.80	8.10	....	11.03	....	16.63

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[1] Bulletin 1, Bureau of Mines. The Volatile Matter of Coal.

volatile product in union with carbon, thus explaining to a great extent the difference between the determined heat value of the coal and that calculated by Dulong's method; Table XIV gives the results of some of these experiments.

With reference to this table, Porter and Ovitz state that there is a possibility of the formation of CO_2 in slight amount from the oxygen of air in contact with the coal at the beginning of a test, and that on the assumption that 500 cc. of air is in contact with the coal, there could be formed, if all its oxygen entered into CO_2, only 0.28 gram of CO_2, or 0.07 per cent., on 400 grams of coal.

Dulong's calculation of heat value from the ultimate analysis assumes that all of the oxygen of the coal combines with the hydrogen of the coal during combustion, thereby neutralizing, so to speak, the calorific value of an amount of hydrogen equal to $O \div 8$.

Porter and Ovitz state that by

> "Combining with carbon instead of with hydrogen in the coal, the oxygen exercises less anticalorific influence on the efficiency of the coal, as 1 gram of oxygen in combining with carbon to CO_2 neutralizes $\frac{3}{8}$ gram of carbon, or 3060 calories; in combining with carbon to CO it partly neutralizes $\frac{3}{4}$ gram of carbon, thus neutralizing $\frac{3}{4} \times 2417 = 1813$ calories; whereas 1 gram of oxygen in combining with hydrogen to H_2O neutralizes $\frac{1}{8}$ gram of hydrogen, or $\frac{1}{8} \times 34{,}460 = 4308$ calories. The anticalorific influence of a unit of oxygen in forming CO_2 or CO is therefore approximately 70 per cent. or 42 per cent., respectively of its influence when forming water."

The investigations of Porter and Ovitz, given in Bulletin 1, Bureau of Mines, contribute a great deal of valuable data on the composition of the volatile products arising from the distillation of various kinds of coals, and as evolved at different temperatures. The main features of these results point to the comparatively large amounts of inert constituents, such as CO_2 and water in the products of certain western coals; the large amounts of higher methane hydrocarbons, such as ethane, in the products produced by the application of moderate temperatures, this being particularly true of the coals from the Appalachian fields; and finally the larger quantities of gas and tarry vapors produced quickly at moderate temperatures during the distillation of the younger western coals. This bulletin also shows that certain bituminous coa [illegible]e western fields are very well adapted to the production of a high-grade illuminating gas with the other by-products of coking, principally ammonia.

Similar investigations to those mentioned above were made by E. Börnstein on the distillation of German bituminous and brown coals at varying temperatures up to 450°C. (842°F.). His experiments resulted in tars with no aromatic hydrocarbons and gases containing large amounts of CO_2 and homologues of methane.

E. J. Constam and E. A. Kolbe investigated the distillation of a number of bituminous coals of varying composition, distillation being carried on under high temperatures, resulting in the production of large amounts of CO_2 and CO from those coals high in oxygen content, but the product contained less hydrocarbons.

L. Vignon distilled five coals, each of a different oxygen content, at 900°C. (1652°F.), and attempted to establish a ratio between the amount of oxides of carbon and the oxygen in the coal.

David White[1] made a comparative study of ultimate coal analyses in order to determine the relative importance of oxygen as an impurity in various coals; to illustrate the transition between various grades of coal of similar origin—transition mainly due to progressive volatilization, brought about more or less directly by dynamic influences; to ascertain the relative proportions of oxygen, hydrogen and carbon in coking coals, especially with reference to a theory tentatively framed to explain the coking quality; and to forecast the probabilities of successfully treating a large number of coals so as to render them amenable, in a softened condition, to satisfactory study under the microscope.

White states that the great variation in the kinds and qualities of coal is due to:

1. The kind and sources of the original ingredient matter.
2. To the conditions of accumulation and deposition of this matter.
3. To the phase and stage of progress of the coal-forming process—the transformation of the organic matter into coals of various grades.

When the organic matter, which consists chiefly of various vegetal types, is accumulated under conditions favorable for the formation of coal, the process of formation goes forward, this process being divided into two stages, viz:

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[1] Bulletin 29, Bureau of Mines. "The Effect of Oxygen in Coal."

1. The putrefaction or fermentation stage, in which the action is essentially biochemical, due to the work of anaerobic bacteria, and which is, geologically speaking, relatively soon terminated.

2. The stage of chemical and physical alteration, the latter being mainly induced and controlled by influences which are geologically dynamic, and which action is termed dynamochemical, this latter action being still in progress, its ultimate goal being to practically devolatilize the coal and to transform it into graphite.

The most important change taking place during both stages is the elimination of oxygen, this being also the most economically essential incident of the transformation. Cellulose and the lignose of the woody matter, forming a great portion of the organic matter of coal, contains over 40 per cent. of oxygen, and this process of deoxygenation can be appreciated when only 2.5 per cent. and less oxygen is found in anthracite; the less the per cent. of oxygen remaining in the coal, the higher will be its heating value.

Renault and **Bertrand** claim, after examining a number of bogheads and oil shales that these substances owe their physical and chemical characters to the presence of large quantities of gelatinous algæ, which in these coals seem to have exercised a selective attraction for certain bituminous compounds, and it is reasonable to admit that these lower organisms exerted quite some influence on the character and quality of the final converted organic residuum.

White states that he is disposed to regard the enriching bitumen as being derived from the decay of associated animal organisms, as well as from the putrefaction products of great quantities of the gelatinous algæ themselves, and that the relatively increasing importance of the bituminous matter in the more mature, older, or more altered condition of the coal is due to concentration as the result of devolatilization and reduction of the coal by the dynamochemical process, the larger part of the concentration being the result of loss of oxygen, which is in larger proportion at the outset.

This loss of oxygen is very great as compared to the loss of hydrogen, and the progressive deoxygenation of the organic matter effects a concentration of the bituminous compounds in the body of the organic material which is being subjected to the process of coal formation, or it is producing bituminization.

WHITE's investigations led to the statement that among coals of the same degree of progressive devolatilization those containing the greatest quantity of recognizable micro-algæ show hydrogen and oxygen in almost the same proportion as exists in bitumen, but that as a general proposition it also seems, conversely, that those normally sedimented coals, the ultimate analyses of which most nearly approach that of bitumen, bear the more distinct microscopical evidence of the presence of the algæ. Those coals, whose large volatile combustible matter contains relatively the highest hydrogen and the lowest oxygen, are those in which the organic remains known as micro-algæ are most predominant. He therefore concludes that coals high in volatile matter and whose analyses show sufficiently high bituminization will coke by the ordinary, or bee-hive process, and that the degree of bituminization in these coals is indicated by the relative excess of hydrogen as compared with the diminished oxygen in dry coal, expressed by the ratio H : O.

During WHITE's investigations he examined and compared the results obtained from the chemical laboratory of the United States Geological Survey on 319 different American coals, and an examination of his tabulation shows that below the highest of the semi-bituminous coals which are approaching the anthracite stage, those coals with a H:O ratio, or percentage, of 59 or more, with but one or two exceptions, make coke by the ordinary commercial process (in bee-hive ovens). Nearly all those below 59 and above 55, so far as tested, make a coke, and among those with a ratio of 50 a large percentage make coke, and one or two of the tested coals, with a slightly lower ratio, also coke, but those cokes made from coals with a H:O ratio less than 55 are usually very poor and apt to be brittle and dark, the best cokes being made from coals in which the H:O ratio is 60 or more.

WHITE's investigations show that pure coals having a fixed carbon percentage of 66 or 67, also have a H:O ratio of 60 per cent. or more, and therefore these coals should be fusible and susceptible to coking. This holds good up to a fixed carbon of about 79 per cent., beyond which he found that some of the coals examined did not coke well by the ordinary process, and some of the others apparently refused to coke at all. Practically all coals, with an available hydrogen content of 3.80 or more, coke well unless this hydrogen content is accompanied by too high a percentage of fixed carbon, and some good cokes are produced from coals in which the available hydrogen is as low as 3.20 per cent.

All of this evidence points to the fact that in coals possessing less than 79 per cent. fixed carbon in the pure coal, the H:O ratio, on a dry coal basis, offers reliable data for determining the coking quality of the coal from the ultimate analysis, and that in the higher coals it is generally possible to produce coke if the quantity of carbon in the volatile matter is relatively large; also that most of the coals in which the H:O ratio is high, but which refuse to coke, are distinguishable by their clearly defined calorific deficiency with reference to the carbon: oxygen + ash ratio.

White therefore concludes that "the adaptability of a coal to coking by the ordinary (bee-hive) process appears to be indicated with a fair degree of certainty by the ratio of the hydrogen to the oxygen, moisture-free basis. Practically all coals with H:O ratios of 59 per cent. or over seem to possess the quality of fusion and swelling necessary to good coking. Most coals with ratios down to 55 will make coke of some kind, while a few coals with ratios as low as 50 coke in the bee-hive oven, though very rarely producing a good article. The coking property seems to depend, not so much on the amount of available hydrogen, which is a very imperfect index of the proportion of the elements in the volatile, as on the relative amount of hydrogen as compared to that of oxygen. In those coals undergoing change to anthracite, the hydrogen-oxygen ratio may fail as a guide; the failures appear, however, to be readily distinguished by the marked calorific deficiencies shown by the C:(O + ash) ratio and efficiency curve. The data examined, though insufficient to serve as a basis for a conclusion, seem to point toward the need of a relatively high carbon element in the remaining volatile combustible of coals with 79 per cent. or more of fixed carbon (pure coal), in order to secure either the best coking results or the calorific efficiency indicated by the C:(O + ash) ratio efficiency curve."

As a practical test for the coking quality of coals, M. Pishel[1] recommends that a small quantity of the coal to be tested be pulverized in an agate mortar until it will pass through a 100-mesh sieve. With some coals the mortar and pestle will be covered with a coating of coal dust which will adhere so strongly to the agate surface that it can be removed only with difficulty, the degree of adhesion seeming to coincide with the coking qualities of the coal. This is not to be confounded with the tendency shown by some coals to pack together, as, though this appears to be limited to coking coal, the packing is probably due more to the moisture content of the coal than to its coking quality.

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[1] Am. Gas Lt. Jour., 1908, page 445.

The result of WHITE's investigations leads him to make suggestions relative to blending various coals in order to make it possible to secure the best coking results, or to treat coals that fail to coke by the ordinary process and thus secure the desired result.

He does not suggest only the mixing of coals according to regional trade conditions, or the mixing of coals possessing relatively high H :O ratios and a moderate fixed carbon with others whose ratios are not up to the required standard for good fusion, but also to mix such coals having a high percentage of fixed carbon with others which are rich in hydrogen and carbon.

The blending or mixing of coals has as its ultimate object the production of coke of suitable quality for various purposes, but proper blending may also lead to increased by-products. Some coals which do not, under ordinary conditions, produce good coke are very rich in by-products, especially ammonia, and at times such coals are found in the same mine with coals of good coking quality, and it is therefore fair to suppose that the proper mixture of these two classes will produce a good coke, and at the same time increase the by-product yield; it must be stated, however, that to accomplish this result a thorough mixing of the two grades is required.

The principal characteristics of coking coals should be, first, a prismoidal structure which will permit of the free mixing of the particles, and a proper diffusion of the gases; and, second, a chemical composition which combines not only the fusing and cementing factors to a favorable degree, but also such desirable elements in both proportion and form as will make the coal adaptable for the production of coke of a desired character. Coals which could not be coked some years ago are now being used to advantage by first subjecting them to crushing and pulverizing, thus breaking up the laminated or crystalline structure which differentiated them from the usual coking coals, this crushing being necessary because, in its original state, the structure of the coal prevented the free mixing of the particles and the diffusion of the gases, thus not permitting the adjustment necessary for the process of fusion, cementation, and the interchange of position of the various elements required by the coking process.

A. DOUGLAS[1] states that it is impossible to convert a non-coking into a coking coal, but that, in some cases, a coal which ordinarily yields a soft, friable coke may be caused to yield a good coke by subjecting it to compression. The coking property cannot be created, but it can be destroyed, because if a sample of good coking coal is dried at 105°C. (221°F.) and then heated in a hot-air oven, the result will be a decrease in weight at first, followed by an increase in weight as the pressure rises. If the coal is heated to 200°C. (392°F.) for about an hour, it will lose its coking quality and, if subjected to the usual crucible test, a black powder will be found in the bottom of the crucible in place of the usual button of coke; therefore, in all cases where coals were submitted to slow oxidation, the coking property was destroyed, which fact is also borne out by the reduction in the coking quality of a coal which has been subjected to the slow oxidation process attending weathering. An experiment recorded by DOUGLAS seems to disprove the theory that the presence of a considerable quantity of ash in the coal will affect its coking qualities; he states that a fresh sample of coal was mixed with 30 per cent. of coal ash and then carbonized, with the result that the coke was quite undistinguishable from that produced from the unadulterated coal. Another mixture consisting of 50 per cent. coal and 50 per cent. fine stone dust was also carbonized, the result being a good coke, but not as hard as that from the original coal. Further experiments were made to determine to what extent a coking coal could be adulterated with a non-coking coal and still be capable of coking; for this purpose a number of samples containing varying proportions of coking and non-coking coal were made up and carbonized under similar conditions. Practically no difference in the quality of the coke was observed with mixtures containing up to 30 per cent. of non-coking coal, but above this figure the difference began to be noticeable, and at 80 per cent. of non-coking coal the resultant coke was very poor and totally unsuitable for practical purposes. These tests were repeated with other coals, the same general results being obtained, but it was impossible to determine any fixed proportions which would apply to mixtures of all kinds, as the amount of each variety would depend in each case upon the extent to which it possesses coking property.

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[1] Gas World, Oct. 5, 1912, page 415.

In his further experiments on the coking property of coals[1] he mentions that tarry constituents are often held responsible for this property, but he found that tarry matter is difficult to oxidize, and that there is no noticeable difference between the coking properties of coals containing much and little tar, but in spite of this he showed experimentally that tar, or, better still, pitch does cause carbonaceous matter to coke. In order to determine to what extent pitch acted as a binder, DOUGLAS mixed a series of coals, running from one of first class quality through poorer grades to one that had been heated until it had lost its coking property, with a pitch of medium hardness, these mixtures being then subjected to the usual crucible test. In the case of the best coal he found but little difference in the resultant coke, but the effect became greater with decreasing quality of coal, and the poorest sample produced a soft coke, this sample, before being mixed with pitch, having only resulted in producing a powder in the crucible. He then increased the percentage of pitch and secured a better quality of coke, but could not bring it to equal the quality produced by the best grade of coal; the next to the poorest grade of coal, however, produced a coke which was almost as good as the best.

DOUGLAS concludes from this that some other medium than pitch operates during coking, and that the property which is destroyed by heating to 200°C. (392°F.) cannot be replaced by the addition of pitch, but, at the same time, he thinks it possible that a coal which possesses the property of coking to only a slight extent, can be rendered capable of producing a good grade of coke by the addition of pitch.

A. L. M'CALLUM[2] investigated the action of organic sulphur in coal during the coking process, and states that to the best of our knowledge, sulphur occurs in coal as (1) sulphates, (2) iron pyrites, and (3) organic sulphur. The coal he investigated was from Nova Scotia, and was practically free from sulphates; so in this case only iron pyrites and organic sulphur had to be considered. M'CALLUM tried to procure a series of samples with a continually decreasing amount of organic and an increasing amount of inorganic sulphur, in order that the action of the organic sulphur during coking might be observed. He obtained this series by floating the coal in solutions of calcium chloride of various densities, thus fractionating by weight, the samples secured being given in Table XV.

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[1] Gas World, Oct. 26, 1912, page 529.

[2] Chemical Engineer, Jan., 1910, page 27.

TABLE XV

Sample	Sp. gr.	Volatile	Fixed C	Ash	Sulphur	S in coke
1	1.323	35.10	59.74	5.16	2.06	1.80
2	1.275	35.92	61.57	2.51	1.29	1.17
3	1.261	36.10	62.27	1.63	1.09	0.85
4	1.253	37.47	61.50	1.04	0.95	0.78
5	1.243	37.75	61.35	0.90	0.88	0.68

The percentage of organic sulphur was determined by finding the amount of iron, assuming that this was combined as FeS_2, deducting the sulphur so found from the total and terming this remainder organic sulphur.

To determine the fixed and volatile sulphur, he first determined the total sulphur in the coal, and then that in the coke, the results obtained being given in Table XVI, this table also showing the percentage of organic and inorganic sulphur obtained as described above.

TABLE XVI

Sample	Organic S, per cent.	Inorganic S, per cent.	Volatile S, per cent.	Fixed S, per cent.
1	37.86	62.14	33.49	66.51
2	56.69	43.41	42.64	57.36
3	71.56	28.44	50.46	49.54
4	83.16	16.84	49.47	50.46
5	85.23	14.77	52.27	47.73

While the fixed and volatile sulphurs do not show the same regularity as do the organic and inorganic, there seems to be an increase in the amount of volatile sulphur in those samples having a high percentage of sulphur. Assuming that the only sulphur volatilized was the one atom in the pyrites, the percentage of volatile sulphur may be calculated, because the sulphur termed inorganic is assumed to be present only in the form of pyrites; consequently one-half of the inorganic sulphur should correspond with the percentage of volatile sulphur, providing the above assumption is true, and in case there is no organic sulphur volatilized. The result of this calculation is given in Table XVII.

TABLE XVII

Sample	One-half inorganic S	Volatile S	Difference
1	31.07	33.49	2.42
2	21.70	42.64	20.94
3	14.22	50.46	36.24
4	8.42	49.47	41.05
5	7.38	52.27	44.89

The data here given would seem to indicate that when the inorganic sulphur is in excess, the above supposition is approximately true, but not otherwise, and it seems rather strange why this should be so unless it is due to mass action. M'CALLUM therefore concludes from his experiments that a very considerable portion of the organic sulphur is volatilized during the process of coking.

HEMPEL and LIERG[1] state that suitable treatment will cause any coal to yield a good coke. They experimented on a BUSCHTEHRADER coal which contained 4.9 per cent. of ash, and which yielded a coke which could be readily crumbled between the fingers, but when mixed in the following part by weight: coal, 120; coke from the same coal, 18; clay, 4.8; commercial coal-tar pitch, 8, the latter being dissolved in 18 parts of tar from the coal in question, the mixture, when heated in a crucible, yielded a coke which had a crushing strength of 5600 lb. per square inch, and which contained but 10 per cent. of ash; their experiments also showed that the best cokes, obtained in the ordinary way, had a crushing strength of from 2050 to 2600 lb. per square inch. The conclusion reached by HEMPEL and LIERG is that the quality of coke can be greatly improved by adding clay, tar, and pitch to the coal.

The UNITED STATES STEEL CORPORATION[2] has had a varied experience in the use of high or low volatile coals, alone or in a mixture, for the purpose of producing a high-grade blast-furnace coke. At Joliet, 60 per cent. of high and 40 per cent. of low volatile coal was used at first, the final mixture determined on being 20 per cent. of high and 80 per cent. of low volatile coals; neither of these mixtures caused any oven trouble, but the latter one produced the best furnace coke. After the ovens had been in operation for some time and the oven crews had become accustomed to their working, it was found that a first-class furnace coke could be produced with a mixture of 40 per cent. high volatile and 60 per cent. low volatile coals. At Farrell, Pa., a mixture of 60 per cent. high and 40 per cent. low volatile coals is being coked, while at Birmingham, Ala., a mixture of medium coals, averaging about 29 per cent. volatile matter, is being coked.

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[1] London Jnl. of Gas Ltg., 1911, page 822.

[2] C. A. MEISSNER, Chairman, Coke Committee.

At Wheeling, W. Va., the National Tube Company, at their Riverside furnaces where the SEMET-SOLVAY ovens are being operated, carbonize a mixture of 80 per cent. high volatile Connellsville coal and 20 per cent. low volatile coal, the mixture averaging 32 per cent. in volatile matter, while at the Zenith furnace, Duluth, a fair-looking coke is being made out of 100 per cent. of Pittsburgh seam, screened coal, which averages 36 per cent. in volatile matter, this coke being used in a small furnace, and it is therefore questionable as to what result would be secured by its use in a large furnace.

MR. MEISSNER further states:

> "That it is not safe to lay down any hard and fast rule as to whether the by-product coke oven demands high or low volatile coal, nor are there any definite proportions that determine the value of the quality of the coke. It is a question of cutting and trying, and above all of having uniform and good coke oven operations and adapting the furnace practice to these conditions. Eternal vigilance and the least possible variation in the operations is the *sine qua non* of successful by-product coke oven operation."

O. L. BOUDOUARD[1] states that in order to determine the nature of the substances which enable certain coals to give a hard, compact coke, he has studied the solvent action of a number of chemical compounds upon a Belgian coal which produced the desired character of coke. He considered only three reagents, namely: nitric acid, concentrated sulphuric acid, and SCHWEITZER'S reagent, stating that the dehydrating action of the concentrated sulphuric acid upon hydrocarbons was sufficiently energetic to completely carbonize them; also that the action of SCHWEITZER's reagent may be foreseen by its solvent action on the materials of cellulose origin contained in the coals.

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[1] Le Genie Civil, Feb. 20, 1909, page 286.

The different organic solvents used made no appreciable modification on the quality of coke given by exhausted coal; also the concentrated sulphuric and nitric acids used caused the coals to lose their coking power, and melted potassium and SCHWEITZER's reagent reduced the coking power to an appreciable degree. Successive action of aqueous potassium, hydrochloric acid, and SCHWEITZER's reagent caused the coking power to disappear completely, BOUDOUARD stating that the explanation of this phenomenon should be sought in the solvent power of the cupropotassic reagent for substances of cellulose origin, and that the humic acid found in the oxidized and non-coking coals arises from the oxidation of these hydrocarbonaceous substances.

He states that anthracites do not produce a compact coke owing to the fact that this variety of carbon, which represents a very advanced stage of transformation of vegetable substances, no longer contains the cellulose or disappearing hydrocarbonaceous matter, and that it is difficult to even find a trace of humic acid in anthracites, whether they be natural or oxidized.

He concludes that the appearance of even a very slight amount of humic acid in a coal will cause its coking quality to disappear immediately.

Two characteristics govern the selection of a gas coal to be used in the production of gas, the first being *physical quality* and the second, *chemical composition*. Under the first caption the gas value will depend upon whether the coal is hard and lumpy or whether it is friable or liable to slack during handling, and it should be as free as possible of such impurities as iron pyrites or slate. In its chemical composition it should show the proper percentage of volatile combustible, and its moisture, sulphur, and ash content should be low; the percentage of ash should be such that the resultant coke would be of commercial character.

A 2000-lb. ton of high-grade gas coal, when properly carbonized in retort benches, should produce the following approximate merchantable products:

	Weight		Per cent.
Coke	1300 lb.	=	72.9
Coal tar, 13 gal	130 lb.	=	7.1
Ammonia (NH_3)	5 lb.	=	0.3
Gas, 10,000 cu. ft	350 lb.	=	19.7
Total	1785 lb.		100.0

the 1300 lb. of coke being 65 per cent. of the 2000 lb. of coal charged.

Using average selling prices, these commercial products have a value of:

	Value	Per cent.
1300 lb. of coke at $4.00 per ton	$2.60	19.4
13 gal. of tar at 3 cts	0.39	2.9
5 lb. of ammonia (NH_3) at 8 cts	0.40	3.0
10,000 cu. ft. of gas at $1.00	10.00	74.7
Total	$13.39	100.0

With this production, and with the values given, the value of the gas to the coke will be approximately as 4 is to 1, and the value ratio of tar to ammonia is approximately 1, these ratios of course varying with the local market prices of the products.

It is impossible to tell what the quality of either gas or coke will be when produced by a coal the proximate analysis of which only is known, as to arrive at any definite conclusion requires an actual carbonization test, and this should be carried out under actual operating conditions. The various coals used also show quite some variations in the production of the other residuals besides coke, and this quite independently of the temperature of carbonization.

The Bureau of Mines[1] has been engaged for some time in determining what coals are available for the manufacture of illuminating gas, the object being to locate coals which are cheaper than the Pennsylvania coals usually used, as there are but a few well-developed coal fields in this country capable of furnishing a coal which will satisfy all of the requirements of the manufacture of illuminating gas; the Bureau's investigations were carried on at the testing station established by the Michigan Gas Association at Ann Arbor, and were under the direct supervision of PROF. ALFRED H. WHITE, of the University of Michigan.

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[1] Bureau of Mines, Bulletin 6, Coals Available for the Manufacture of Illuminating Gas.

The coals tested were from eleven different localities, as follows:

TABLE XVIII.—SOURCES OF COAL TESTED AT ANN ARBOR

No.	Bed	Locality	Railroad
1	No. 2 or lower	Hanna, Carbon County, Wyo.	Union Pacific.
2	Saginaw	Saginaw, Saginaw Co., Mich.	Michigan Central.
3	Yampa	Oak Creek, Routt Co., Colo.	Denver, Northwestern and Pacific.
4	No. 5	Harrisburg, Saline Co., Ill.	Big Four.
5	Rex	La Follette, Campbell County, Tenn.	Louisville and Nashville.
6	Pittsburgh	Scott Haven, Allegheny County, Pa.	Pittsburgh and Lake Erie.
7	Upper Elkhorn	Hellier, Pike County, Ky.	Chesapeake and Ohio
8	Raton	Van Houten, Colfax County, N. Mex.	Santa Fe.
9	Thompson	Blockton, Bibb County, Ala.	Mobile and Ohio.
10	Kanawha, No. 2.	Page, Fayette County, W. Va.	Virginian.
11	Sopris	Sopris, Las Animas Co., Colo.	Colorado and Southern.

While comprehensive tests were made on these coals, the result of this work is to be considered only a progress report, and the results are to be taken as tentative and suggestive, and not in any way final; this is due to the fact that any given coal which will produce good results in one gas works, may prove very unsatisfactory in another, owing to prevailing conditions at the latter, therefore it should not be expected that the statements issuing from a testing station would be final as to the value of a given coal.

The report states that

> "Gas coals should be tested in the retort at both high and low temperatures, and the tests should be repeated until it is fairly certain that no serious error has been made; but since a long time is required for a complete study of a number of coals, it has seemed wise to present the data collected on these eleven coals."

The authors state that after studying the results they feel that while these results suggest some interesting deductions,

> "The only conclusion which seems to apply to all coals well enough to warrant presentation is that relating to the percentage yield of coke."

The percentage of coke yield is not mainly dependent on retort temperature, as in all cases the retort is sufficiently hot to drive off the volatile matter contained in the coal and besides this, the secondary changes of the products of distillation at high temperatures in the retorts do not affect the yield of coke to any appreciable degree.

The expected coke yield was taken as being roughly proportionate to the ratio of total fixed carbon to the volatile constituent of the coal, and from calculations based on

$$\frac{\text{Fixed carbon} + \text{ash}}{\text{Moisture} + \text{volatile matter}}$$

the coal was given rank as shown in Table XIX; an examination of the table shows that the computed ratio as per the above agrees closely with the actual percentage of coke produced, an error in one of two tests with coal from New Mexico being possibly responsible for the difference shown with this coal.

TABLE XIX.—COKE YIELD OF COALS TESTED

Rank of coal	$\frac{\text{Fixed carbon} + \text{ash}}{\text{Moisture} + \text{volatile matter}}$	Percentage of coke from coal as charged
1. Wyoming	0.82	50.2
2. Michigan	1.43	59.5
3. Oak Creek, Colo	1.53	60.0
4. Illinois	1.55	62.3
5. Tennessee	1.58	66.8
6. Pennsylvania	1.82	67.0
7. Kentucky	1.88	67.0
8 New Mexico	1.93	68.9
9. Alabama	2.04	68.3
10. West Virginia	2.21	73.3
11. Sopris, Colo	2.90	74.9

The average results secured on tests with the Pittsburgh coal were:

Coke, per cent. of coal charged	67.07
Gas, per pound of coal charged (cubic feet)	5.04
Candle power, approximate average	15.50
Candle feet per pound of coal	79.50
Heating value per cubic foot (B.t.u.)	641.00
Heating value per pound of coal (B.t.u.)	3280.00
Gas analysis (per cent.):	
Carbon dioxide	1.30
Illuminants	3.70
Oxygen	0.80
Carbon monoxide	6.50
Methane	34.40
Hydrogen	48.20
Nitrogen	5.00
Ammonia (NH_3) per ton of coal charged (pound)	5.43
Tar per ton of coal charged (pounds)	155.80

TABLE XX.—COAL AS CHARGED

Coal bed	Proximate analysis					Heat value, B.t.u.
	Moisture	Volatile matter	Fixed carbon	Ash	Sulphur	
Alabama	2.71	29.13	63.87	4.29	0.50	13,990
Oak Creek	7.17	32.36	45.92	14.55	1.00	10,953
Sopris	1.70	23.90	56.26	18.14	0.57	12,312
Harrisburg, Ill.	4.66	34.44	53.71	7.19	1.96	12,919
Hellier	2.46	31.18	60.11	6.25	0.43	13,885
Hellier	3.17	32.40	60.67	3.76	0.45	14,200
Saginaw	9.28	31.67	53.70	5.35	0.98	12,456
Van Houten	2.23	31.42	51.25	15.10	0.67	12,438
Van Houten	2.32	32.18	56.56	8.94	0.70	13,385
Scott Haven	1.92	32.82	58.85	6.41	1.12	14,026
Scott Haven	2.18	32.96	57.33	7.53	1.43	13,815
Scott Haven	2.05	33.25	58.02	6.68	1.49	13,955
Scott Haven	2.43	32.70	59.99	4.88	0.85	14,036
LaFollette	3.13	34.99	59.73	2.15	0.72	14,245
Page	1.28	30.65	62.50	5.57	1.53	14,531
Page	1.27	29.02	62.88	6.83	1.18	14,344
Wyoming	22.56	32.26	39.37	5.81	0.36	9,592

TABLE XXI.—DRY COAL

Coal bed	Calculated proximate composition				Heat value, B.t.u.	Ultimate analyses[1]					
	Volatile matter	Fixed carbon	Ash	Sulphur		Hydrogen	Carbon	Nitrogen	Oxygen	Sulphur	Ash
Alabama	29.94	65.65	4.41	0.51	14,380	5.35	80.50	1.24	8.21	0.62	4.08
Oak Creek	34.86	49.47	15.67	1.08	11,799	4.66	70.41	1.59	12.44	1.58	9.32
Sopris	24.31	57.24	18.45	0.58	12,524	4.48	72.83	1.07	6.46	0.66	14.50
Harrisburg	36.13	56.33	7.54	2.06	13,550	4.89	76.21	1.43	7.93	1.77	7.77
Hellier	31.97	61.62	6.41	0.44	14,234	5.01	79.73	1.22	7.56	0.55	5.93
Hellier	33.46	62.66	3.88	0.46	14,665	....	.....	....	.....	....	.....
Saginaw	34.91	59.19	5.90	1.08	13,730	5.13	75.56	1.35	8.79	1.41	7.76
Van Houten	32.14	52.42	15.44	0.69	12,722	5.08	73.68	1.28	6.01	1.22	12.73
Van Houten	32.95	57.90	9.15	0.92	13,703	....	.....	....	.....	....	.....
Scott Haven	33.46	60.00	6.54	1.14	14,301	5.24	79.00	1.38	6.72	1.14	6.52
Scott Haven	33.70	58.60	7.70	1.46	14,123	....	.....	....	.....	....	.....
Scott Haven	33.94	59.24	6.82	1.52	14,247	....	.....	....	.....	....	.....
Scott Haven	33.51	61.49	5.00	0.87	14,386	....	.....	....	.....	....	.....
La Follette	36.12	61.66	2.22	0.74	14,706	5.00	78.14	1.67	8.30	1.83	5.06
Page	21.05	63.31	5.64	1.55	14,720	5.09	82.46	1.42	4.95	0.83	5.25
Page	29.39	63.69	6.92	1.20	14,530	....	.....	....	.....	....	.....
Wyoming	41.66	50.84	7.50	0.46	12,386	5.03	68.80	1.08	17.58	0.42	7.09

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[1] The ultimate analyses are mostly of the mine samples, and hence differ from analyses of test samples, especially in the percentages of ash. The composition of the coal free from moisture and ash probably *does not differ much*.

TABLE XXII.—COMPOSITION OF COALS

Coal bed	Calculated proximate composition			Heat value, B.t.u.	Calculated ultimate composition				
	Volatile matter	Fixed carbon	Sulphur		Hydrogen	Carbon	Nitrogen	Oxygen	Sulphur
Alabama	31.32	68.68	0.53	15,043	5.58	83.92	1.29	8.56	0.65
Oak Creek	41.34	58.66	1.28	13,991	5.14	77.65	1.75	13.72	1.74
Sopris	28.81	70.19	0.71	15,358	5.24	85.18	1.25	7.56	0.77
Harrisburg	39.07	60.93	2.23	14,656	5.30	82.63	1.55	8.60	1.92
Hellier	34.16	65.84	0.47	15,210	5.32	84.75	1.30	8.05	0.58
Hellier	34.81	65.19	0.48	15,257	......	......	......	......	......
Saginaw	37.10	62.90	1.15	14,591	5.56	81.91	1.46	9.54	1.53
Van Houten	38.01	61.99	0.82	15,046	5.82	84.43	1.47	6.88	1.40
Van Houten	36.27	63.73	0.79	15,082	......	......	......	......	......
Scott Haven	35.80	64.20	1.22	15,302	5.61	84.51	1.48	7.18	1.22
Scott Haven	36.51	63.49	1.58	15,300	......	......	......	......	......
Scott Haven	36.42	63.58	1.63	15,289	......	......	......	......	......
Scott Haven	35.27	64.73	0.92	15,192	......	......	......	......	......
La Follette	36.94	63.06	0.76	15,039	5.27	82.30	1.76	8.74	1.93
Page	32.91	67.09	1.64	15,601	5.37	87.03	1.50	5.22	0.88
Page	31.57	68.43	1.29	15,610	......	......	......	......	......
Wyoming	45.04	54.96	0.50	13,390	5.41	78.05	1.16	18.93	0.45

TABLE XXIII.—RETORT OPERATION, COAL AND COKE

Coal bed	Coal				Coke			
	Per cent. rejected by ¾-in. bar screen	Screened, as charged (lb.)	Charged dry (lb.)	Charged, ash and moisture free (lb.)	Yield (lb.)	Dry (per cent. of coal charged)	Dry (per cent. of dry coal charged)	Per cent. of breeze through ¾-in. bar screen
Alabama	23.40	400	389	372	273	68.3	70.2	12.5
Oak Creek	39.50	400	371	313	240	60.0	64.6	37.3
Sopris	13.83	410	403	329	307	74.9	76.0	10.0
Harrisburg	27.90	400	381	353	249	62.3	65.4	6.7
Hellier	30.80[1]	360	351	329	250	69.4	71.3	....
Hellier	48.50	400	388	372	258	64.5	66.5	9.1
Saginaw	9.70	400	363	341	238	59.5	65.6	17.9
Van Houten	11.95	398	389	329	270	67.8	69.4	13.1
Van Houten	21.90	400	391	355	280	70.0	71.0	11.7
Scott Haven	Screened at mine	404	396	320	271	67.1	68.5	5.0
Scott Haven		400	391	370	270	67.5	69.0	....
Scott Haven		400	392	365	259	64.8	66.1	....
Scott Haven	17.40	400	390	370	274	68.5	70.3	7.6
La Follette	8.37	400	387	379	267	66.8	69.0	7.6
Page	63.27	400	395	372	302	75.5	76.5	10.0
Page	70.00	400	394	368	285	71.2	72.4	7.6
Wyoming	10.40	400	309	286	201	50.2	64.9	88.4

[1] A ½-in. bar screen was used in this test.

TABLE XXIV.—YIELD, HEATING VALUE, AND CANDLE POWER OF THE GAS

Coal bed	Yield of gas per pound of coal (cu. ft.)			Calculated average heating value (B.t.u.)		Calculated average candle power	Calculated average gas analysis (per cent.)						
	Coal as charged	Dry coal	Coal, ash and moisture free	Gross	Net		CO_2	C_nH_{2n}	O_2	CO	CH_4	H_2	N_2
Alabama.....	5.2	5.3	5.6	598	538	14.0	2.7	3.7	1.0	8.9	31.2	49.1	3.4
Oak Creek....	4.8	5.2	6.1	626	566	14.0	4.4	5.3	0.8	11.3	29.8	43.9	4.5
Sopris........	4.9	5.0	6.1	614	550	12.0	1.3	4.2	0.9	6.0	29.5	55.8	2.3
Harrisburg...	4.3	4.6	4.9	632	568	15.2	2.5	3.7	0.8	7.9	33.5	45.6	6.0
Hellier.......	4.8	4.9	5.3	650	578	14.5	1.6	4.3	1.1	6.4	33.1	50.3	3.2
Hellier.......	5.0	5.2	5.4	622	561	14.1	1.9	4.1	0.7	7.0	29.4	51.3	5.6
Saginaw......	4.3	4.7	5.0	593	526	17.4	2.9	5.3	0.7	9.2	34.1	43.1	4.7
Van Houten...	4.5	4.7	5.5	674	606	16.5	2.0	5.9	0.8	8.2	34.8	45.9	2.4
Van Houten...	4.9	5.0	5.5	682	618	16.8	2.1	6.1	0.6	8.2	35.5	45.7	1.8
Scott Haven..	4.8	4.9	6.1	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Scott Haven..	4.9	5.0	5.3	649	593	15.2	1.2	4.4	1.1	6.6	37.9	42.2	6.6
Scott Haven..	5.3	5.4	5.8	611	550	16.0	1.1	3.0	0.8	6.5	31.8	51.6	5.2
Scott Haven..	5.1	5.3	5.6	664	603	15.4	1.7	3.8	0.5	6.5	33.6	50.7	3.2
La Follette...	5.5	5.7	5.8	641	575	15.9	2.0	5.0	0.9	8.2	31.2	49.1	3.6
Page.........	5.2	5.3	5.6	617	555	15.8	1.3	3.8	0.9	4.9	32.9	56.6	2.6
Page.........	5.0	5.1	5.4	622	557	16.1	1.2	3.3	0.6	5.0	32.6	55.1	1.8
Wyoming....	5.4	7.0	7.5	564	502	8.8	7.6	4.9	0.2	14.4	29.0	40.2	3.7

TABLE XXV.—YIELD OF TAR, AMMONIA, SULPHUR AND NAPHTHALENE

Coal bed	Tar per ton of coal (gal.)			Ammonia (NH_3) per ton of coal (lb.)			H_2S per 100 cu. ft. of gas at outlet of scrubber (gr.)	Total naphthalene per ton of coal as charged (lb.)
	Coal as charged	Dry coal	Coal, ash and moisture free	Coal as charged	Dry coal	Coal, ash and moisture free		
Alabama......	19.2	19.7	20.6	4.03	4.15	4.33	....	10.7
Oak Creek....	15.6	16.8	19.9	7.60	8.20	9.60	....	....
Sopris........	14.2	14.5	17.7	4.36	4.43	5.43	....	....
Harrisburg....	17.2	18.1	19.5	6.38	6.70	7.20	1390	4.5
Hellier........	10.8	11.1	11.8	4.01	4.12	4.39	....	....
Hellier........	18.8	19.5	20.2	3.48	3.61	4.25	823	5.7
Saginaw......	16.5	18.2	19.3	6.40	7.05	7.50	3750	5.5
Van Houten...	16.8	17.2	20.3	4.97	5.09	6.02	....	....
Van Houten...	21.8	22.3	24.5	4.32	4.42	4.82	....	....
Scott Haven...	.....	.....	.....	.....	.....	.....	....	....
Scott Haven...	10.7	10.9	11.8	5.80	5.93	6.43	....	....
Scott Haven...	.....	.....	.....	5.18	5.28	5.67	....	....
Scott Haven...	21.1	21.6	22.8	5.33	5.46	5.75	830	8.0
La Follette....	22.0	22.7	23.2	7.39	7.63	7.81	296	....
Page.........	20.1	20.4	21.7	5.19	5.26	5.57	....	....
Page.........	14.6	14.8	15.9	4.37	4.43	4.75	....	11.4
Wyoming.....	14.4	18.6	20.1	3.77	4.88	5.28	1660	....

Tables XX, XXI, XXII, XXIII, XXIV, and XXV show the proximate analysis of the coal as charged; analysis of the dry coal; composition of the coals; retort operation; products of distillation; yields, heating value, and candle power of the gas; and the yield of tar, ammonia, sulphur, and naphthalene, respectively, for the coals from the eleven beds given in Table XVIII.

The data given in these tables are shown graphically in Diagram I,[1] the results being presented at half-hour intervals. For each of ten of the coal beds the diagram shows the temperatures both inside and outside of the retort, the candle power as well as the heating value of the gas, and the variations in the percentages of such important constituents of the gas as illuminants, hydrogen and methane. It will be noted that the quantity of carbon dioxide, carbon monoxide, oxygen, and nitrogen in the gas has not been tabulated, these constituents varying but little during the period of distillation. Besides the above, Diagram I shows the total yield of tar and ammonia liquor at intervals of ½ hour, this also being true for the yield of ammonia (NH_3) as well as the weight of naphthalene vaporized in the gas and later dissolved in the tar at the outlet of the tar extractor. The authors state that "this weight of naphthalene was chosen for presentation because the naphthalene that gets past the tar separator, either vaporized or dissolved, causes trouble, as dissolved naphthalene is set free by the ammonia liquor in the scrubbers."

Elkhorn run of mine coal, after a number of analyses made by various gas companies, showed the following average products:

Content	Coal	Coke
Yield		64.70 per cent.
Moisture	1.65	1.29 per cent.
Volatile matter	35.57	2.40 per cent.
Fixed carbon	59.12	87.91 per cent.
Ash	3.67	7.92 per cent.
Sulphur	0.61	0.47 per cent.
B.t.u.	14,545	13,694

Tar

Yield per ton, in gallons	14.15
Per cent. of free carbon	16.14

Ammonia

NH_3, pounds per ton of 2000 lb.	5.00

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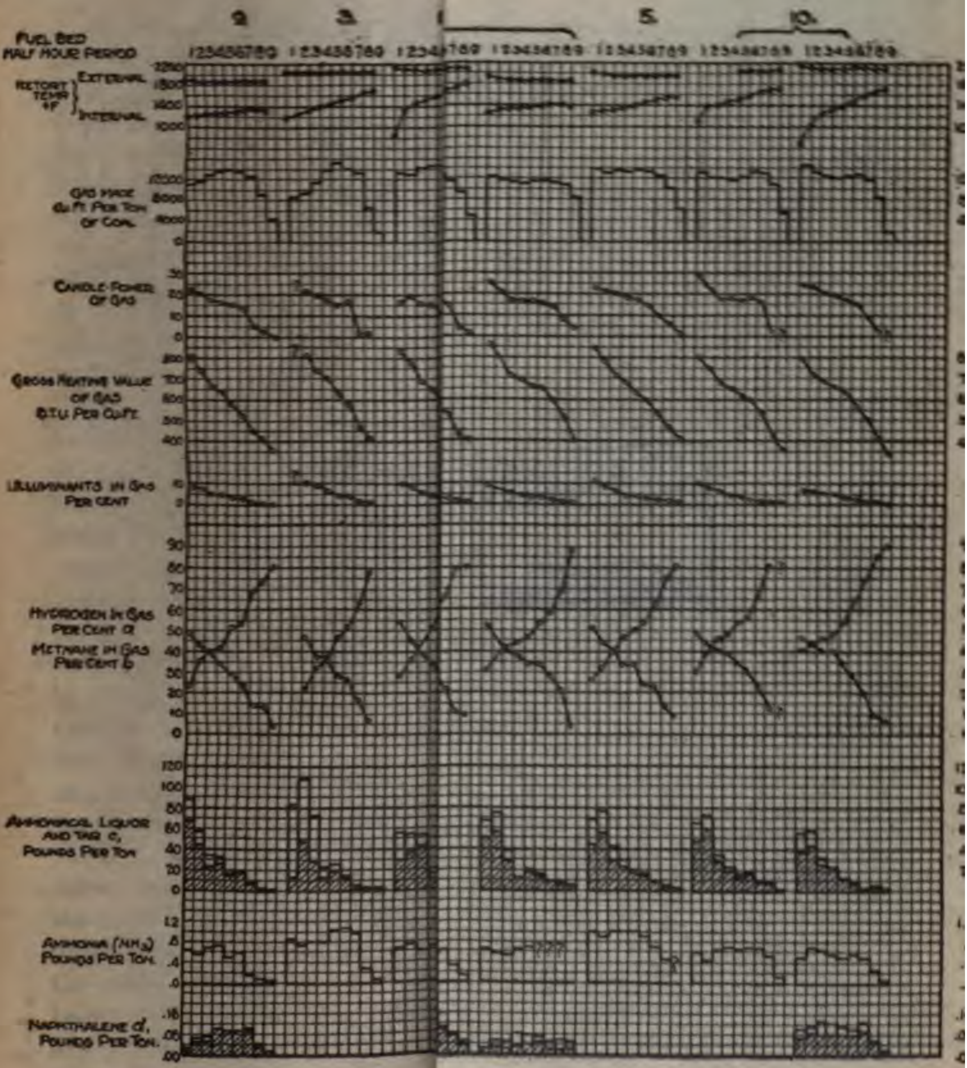
[1] Bulletin 6, Bureau of Mines, A. H. WHITE and PERRY BARKER.



Area Represents Ammoniacal Liquor.

In Tar ; Blank Area Represents -

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THE NEW YORK
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Gas	
Cubic feet per pound	5.34
Candle power	15.25
B.t.u.'s per cubic foot	640.00

DR. KARL BUNTE[1] states that gas coals are those lying between coking coals, which are poor in gas, and the coals which are rich in gas but which coke poorly, and that the proportion of carbon on the dry and ash-free basis averages from 81 to 87 per cent., the calorific power ranging from 13,950 to 14,760 B.t.u. per pound. The most valuable gas making coal is the one which will return the greatest proportion of its calorific value in the form of gas, this proportion varying from 30 to 90 per cent. in the different coals examined at the experimental works of the German Gas Association, the coke containing from 50 to 60 per cent. of the calorific value of the coal, and from 5 to 8 per cent. being found in the tar.

DR. BUNTE further states that there is available no absolute measure of the strength of gas coke, but that the means devised at the experimental station for this determination usually answer all purposes; the cold coke is dropped from the charging stage to the concrete furnace floor, about 11½ ft. below, after which it is screened to four sizes, the results after repeated tests on the same coke being found fairly uniform and reproducible. The system used for determining the value of the coke is based upon the amount of coke passing through a 2½-in. diameter hole in the screen, this screened yield, for the coals examined, varying from 30 to 70 per cent., but only Saar coals were examined under this method; therefore the isolated examples of coals tested do not give any general conclusion as regards coke from these latter classes of coals.

The length of time required to carbonize a coal gives the value of one of its most essential properties, because the longer the period of carbonization, the greater will be the expense as regards carbonizing space and fuel used, and the difference in the coals tested was very notable in this particular, one coal being carbonized in about 4½ hours, while others required from 5½ to 6 hours. Generally speaking, the older coals gasify most slowly, and the more recent coals most rapidly, the time required for carbonization, however, depending to a considerable extent on the size of the coal, small coal requiring a longer period than large or lump coal, consequently with the latt the gas is quickly liberated and is therefore better, as it is su jected to decomposition to a lesser extent; besides this, the g yield from very finely broken coal, even though it be thorough carbonized, is less than the yield from large coal.

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[1] London Journal of Gas Lighting, 1910, page 957.

Burgess and **Wheeler**[1] made an examination of coals f volatile constituents, and their preliminary tests showed that all coals there is a well-defined decomposition point which li between 700° and 800°C. (1292° and 1472°F.), this conditi corresponding with a marked increase in the quantity of hydrog evolved. In carbonizing bituminous coals, the increase in t quantity of hydrogen evolved decreases at temperatures abo 900°C. (1652°F.), but with anthracite there is no decrease up 1100°C. (2012°F.). The evolution of hydrocarbons of the para fine series practically ceases entirely above 700°C. (1292°F while ethane, propane and butane, and probably other high members of the paraffine series form a large percentage of t gases evolved below 450°C. (842°F.).

These investigators further state that as a rule the analysis coal gases as evolved at the different stages of carbonization do not admit of complete differentiation between influence of ti and temperature, because the gases obtained from coal at any o time are secured at different temperatures; therefore, **Burge** and **Wheeler**, in their experiments, employed charges of 2 gra of finely powdered coal, since with such small quantities no gre interval of time elapses before the entire mass attains the te perature of the retort. These investigations lead to the furth statement that coal appears to contain two types of compou of different degrees of ease of decomposition, the more unstab compound yielding the paraffines, while that which is decompos with greater difficulty yields hydrogen only, or possibly hydrog and the oxides of carbon, this view being confirmed by a fraction distillation of the samples in a vacuum. Prolonged exhausti at a low temperature admits of the complete removal of the para fine-yielding constituents, while a compound remains whic on decomposition at a high temperature, yields hydrogen; ther fore the difference between one coal and another may very probab be determined by the proportion of these two types of compoun in the coal. The variation in the amount of carbon monoxi evolved during carbonization is probably due to the decompos tion or interaction of oxygenated compounds in the coal, the quantity of carbon monoxide evolved also increasing with the temperature of distillation. The percentage in the gas although not the absolute quantity, increases to a marked degree towards the end of distillation at a uniform temperature, the percentage increase being due to a falling off in the evolution of the other gases. The formation of carbon monoxide, however, increases with an increase in temperature, hence it may be concluded that the carbon monoxide results mainly from the interaction of oxygenated compounds with carbon from the coal, and not from the direct decomposition of these compounds. The possibility is that it arises through elimination of water from hydroxy compounds, with a subsequent reaction of the water vapor with the carbon.

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[1] London Journal of Gas Lighting, 1911, page 667.

By extracting the sample of bituminous coal with pyridine, a black, porous, coke-like substance was obtained, which, upon distillation at 900° C. (1652° F.), yielded chiefly hydrogen, carbon monoxide, and carbon dioxide, the same as does cellulose, and the percentage of hydrogen increased with the temperature of distillation, causing a conclusion that the more important of the two compounds mentioned above is a degradation product of cellulose. This property BURGESS and WHEELER identify with the "hydrogen-yielding constituents" of coal disclosed by their experiments, and it is probably derived from the resins and gums originally contained in the sap of the coal plant and form the "cement" of a conglomerate of which the cellulose derivatives are the base.

CHAPTER IV

ANALYZING COAL

As an incorrect analysis of the coal may lead to serious difficulties in an attempt to secure desired results, it is deemed proper to quote freely the method adopted by the *Bureau of Mines;* this method was originally recommended by a committee of the *American Chemical Society*, and was adopted by the Bureau with such modifications as experience showed advisable, the development of the method being principally due to E. E. SOMERMEIER, professor of metallurgy at the Ohio State University, aided by valuable suggestions from N. W. LORD, late director of the School of Mines at the same university.[1]

The coal samples are received at the laboratory in galvanized iron cans containing, when full, about 3 lb. of coal, each can being provided with a close fitting screw cap, made practically moisture-tight by being wrapped with a piece of electrician's tape around the joint between the top of the can and the cap. This sample is given a laboratory number, the coal being removed from the can and placed in a tin pan about 9½ in. in diameter and 1¼ in. deep, the accurate weight of the pan being known and recorded. This pan with its sample of coal is next placed in a large drying oven, Fig. 1, in which a temperature of from 30 to 35°C. (86 to 95°F.) is maintained, a current of warm air being drawn through the oven by means of an electric fan.

The sample is thus dried until the loss in weight between two successive weighings, made 6 to 12 hours apart, does not exceed 0.5 per cent., the sample being treated in this manner in order to reduce its moisture to such a condition that rapid change in weight will not occur while the coal is being handled in the course of analyses. The Bureau does not consider the loss of weight by air-drying as an accurate determination, and the "air-drying loss" reported in an analysis simply means that the sample lost the stated percentage of moisture before it came to a condition of moisture equilibrium with the air of the room, at a temperature of 30 to 35°C. (86 to 95°F.).

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[1] Methods of Analyzing Coal and Coke, Tech. Paper 8, Bureau of Mines, F. M. STANTON and A. C. FIELDNER; also Notes on the Sampling and Analyses of Coal, Tech. Paper 76, Bureau of Mines, A. C. FIELDNER.

After the sample of coal has been thus air-dried in this oven, and after its last weighing, it is immediately pulverized to a 10-mesh, and finer, size by being passed through a roll crusher. It is then thoroughly mixed, after which it is reduced with a riffle sampler to 500 grams. This 500-gram portion is now transferred to the porcelain jar of an Abbe ball-mill, the jar being sealed air-tight, and the coal ground down to 60-mesh. The jar of this ball-mill is about one-third full of well-rounded flint pebbles, 1 in. in diameter, and it is rotated at about 60 revolutions per minute. It is stated that the wear of the pebbles and interior of the mill does not increase the ash content to any appreciable extent. Bituminous coals require from ¼ to ½ hour, and anthracites from 1 to 3 hours of rotation to become pulverized to 60-mesh size. When the coal has been properly pulverized, it is poured onto a ½-in. screen and there separated from the pebbles by a vigorous shaking of the screen, the sample being then reduced to about 60 grams by passing it three times through a small riffle sampler. This portion is now quickly passed through a 60-mesh sieve, after which it is placed in a wide-mouthed, rubber-stoppered, 4-oz. bottle, the bottle being numbered and dated, and the total time elasping from the opening of the porcelain jar to the stoppering of the laboratory sampling bottle

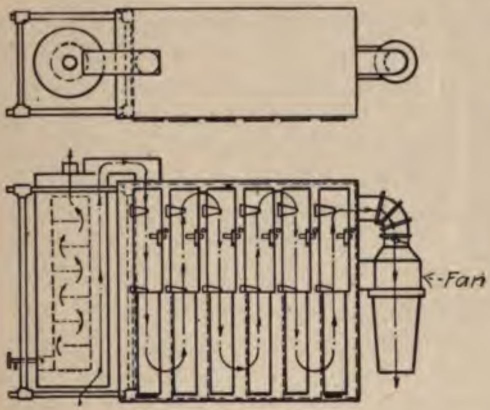

FIG. 1.—Drying oven.

should not exceed 3 minutes; usually from 0.2 to 0.4 per cent. of the sample, in the shape of coarse particles, remains on the sieve, and as these particles are very apt to consist to a large extent of ash-forming substances, they are reduced to 60-mesh on a bucking board and then added to that portion which passed through the sieve.

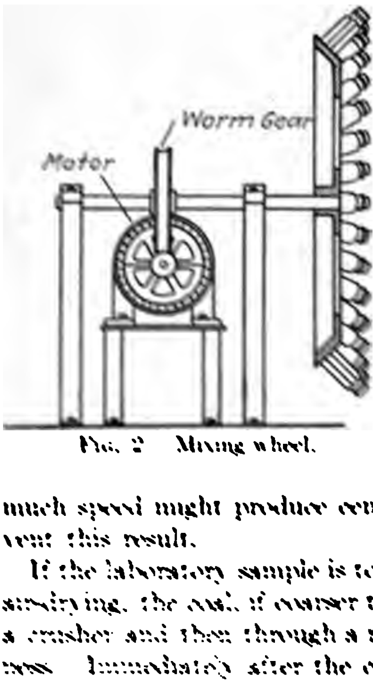

FIG. 2.—Mixing wheel.

This sample is now thoroughly mixed before portions are weighed for the various analytical determinations, the bottle containing the sample being placed on the periphery of the mixing wheel, Fig. 2, and rotated for about 20 minutes. The sampling wheel has a diameter of 30 in., revolving twenty-five times per minute, the periphery being provided with spring clips which hold the sample bottles at an angle of 45 degrees with the axis. These bottles should not be filled to more than one-half their capacity, and rotation of the wheel should be slow enough to ensure a thorough mixing of the sample, as too much speed might produce centrifugal action which would prevent this result.

If the laboratory sample is to be prepared without preliminary air-drying, the coal, if coarser than ¼ in., is first passed through a crusher and then through a mill, thus securing a 20-mesh fineness. Immediately after the coal is received from the 20-mesh mill a 60- to 100-gram portion is taken for the determination of total moisture and is transferred to a rubber-stoppered 4-oz. bottle, the remainder of the sample being thoroughly mixed and then repeatedly passed through a riffle sampler until it is reduced to about 150 grams. This 150-gram portion is now ground to 60-mesh size by means of a planetary-disk pulverizer, again mixed and then reduced with a riffle sampler to about 60 grams, the final portion being placed in a numbered and dated [illegible] The moisture is determined at [illegible] the 60-mesh sample and [illegible] sample. The analysis of the

60-mesh coal, which has become partly air-dried during sampling, is calculated to the dry basis by dividing each result by 1 minus its content of moisture. The analysis of the coal "as received" is computed from the "dry coal" analysis by multiplying by 1 minus the total moisture found in the 20-mesh sample.

As regards these two methods of sampling, it is claimed that the first method is preferable for preparing such laboratory samples as are intended for very accurate analyses, and that the loss of moisture which is unavoidable during sampling is less than by the second method, this being especially so in the case of wet or freshly mined coal samples, because such samples rapidly lose moisture on exposure to air. It is recommended that air-drying be not unnecessarily prolonged as if this should be the case an appreciable loss in heating value will take place through oxidation.

The second method permits of handling a large number of samples in a short time, but the moisture thus obtained is usually somewhat less than that obtained by the first method. In coals which have lost a portion of their moisture content by exposure to the atmosphere, as is usually the case with commercial shipments, this difference need not exceed 0.5 per cent. Samples which are received in a wet condition must be partly air-dried. It is also stated that the disk pulverizer is not adapted to the fine grinding of anthracite, as the coal is heated by the rubbing surfaces to a degree that may possibly change the composition of the sample.

Method of Calculating and Reporting Analyses.—As all the analytical determinations are made on the air-dried sample, the analysis of the coal as received at the laboratory is calculated from the analysis of the air-dried coal in the following manner:

CALCULATIONS FROM ANALYSIS OF "AIR-DRIED" COAL TO COAL "AS RECEIVED"

$$\text{Moisture[1]} \times \frac{(100 - \text{air-drying loss[1]})}{100} \text{ plus air-drying loss[1]} = \text{moisture[1] ``as received''} \qquad (1)$$

$$\text{Volatile matter} \times \frac{(100 - \text{air-drying loss})}{100} = \text{volatile matter ``as received''} \qquad (2)$$

$$\text{Fixed carbon} \times \frac{(100 - \text{air-drying loss})}{100} = \text{fixed carbon ``as received''} \qquad (3)$$

---

[1] All figures expressed in per cent.

$$\text{Ash} \times \frac{(100 - \text{air-drying loss})}{100} = \text{ash "as received"} \qquad (4)$$

$$\text{Sulphur} \times \frac{(100 - \text{air-drying loss})}{100} = \text{sulphur "as received"} \qquad (5)$$

$$\text{Hydrogen} \times \frac{(100 - \text{air-drying loss})}{100} + \tfrac{1}{9}\ \text{air-drying loss} = \text{hydrogen "as received"} \qquad (6)$$

$$\text{Carbon} \times \frac{(100 - \text{air-drying loss})}{100} = \text{carbon "as received"} \qquad (7)$$

$$\text{Nitrogen} \times \frac{(100 - \text{air-drying loss})}{100} = \text{nitrogen "as received"} \qquad (8)$$

$$\text{Oxygen} \times \frac{(100 - \text{air-drying loss})}{100} + \tfrac{8}{9}\ \text{air-drying loss} = \text{oxygen "as received"} \qquad (9)$$

$$\text{Calories} \times \frac{(100 - \text{air-drying loss})}{100} = \text{calories "as received"} \qquad (10)$$

A statement of analysis is referred to a moisture-free basis by calculation from the "air dry" or "as received" analysis in the following manner:

CALCULATION FROM "AIR-DRY" OR "AS RECEIVED" ANALYSIS TO "DRY COAL"

$$\text{Volatile matter} \times \frac{100}{100 - \text{moisture}} = \text{volatile matter in "dry coal"} \qquad (11)$$

$$\text{Fixed carbon} \times \frac{100}{100 - \text{moisture}} = \text{fixed carbon in "dry coal"} \qquad (12)$$

$$\text{Ash} \times \frac{100}{100 - \text{moisture}} = \text{ash in "dry coal"} \qquad (13)$$

$$\text{Sulphur} \times \frac{100}{100 - \text{moisture}} = \text{sulphur in "dry coal"} \qquad (14)$$

$$(\text{Hydrogen} - \tfrac{1}{9}\ \text{moisture}) \times \frac{100}{100 - \text{moisture}} = \text{hydrogen in "dry coal"} \qquad (15)$$

$$\text{Carbon} \times \frac{100}{100 - \text{moisture}} = \text{carbon in "dry coal"} \qquad (16)$$

$$\text{Nitrogen} \times \frac{100}{100 - \text{moisture}} = \text{nitrogen in "dry coal"} \qquad (17)$$

$$(\text{Oxygen} - \tfrac{8}{9}\ \text{moisture}) \times \frac{100}{100 - \text{moisture}} = \text{oxygen in ``dry coal''} \qquad (18)$$

$$\text{Calories} \times \frac{100}{100 - \text{moisture}} = \text{calories in ``dry coal''} \qquad (19)$$

For making comparisons, coal analyses are sometimes referred to "a moisture- and ash-free" basis, by calculation from the "air-dry," "as received," or "moisture-free" analyses, in the following manner:

CALCULATION FROM "AIR-DRY," "AS RECEIVED," OR "MOISTURE-FREE" ANALYSIS TO "MOISTURE- AND ASH-FREE" BASIS

$$\text{Volatile matter} \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{volatile matter referred to ``moisture- and ash-free'' basis} \qquad (20)$$

$$\text{Fixed carbon} \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{fixed carbon referred to ``moisture- and ash-free'' basis} \qquad (21)$$

$$\text{Sulphur} \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{sulphur referred to ``moisture- and ash-free'' basis} \qquad (22)$$

$$(\text{Hydrogen} - \tfrac{1}{9}\ \text{moisture}) \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{hydrogen referred to ``moisture- and ash-free'' basis} \qquad (23)$$

$$\text{Carbon} \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{carbon referred to ``moisture- and ash-free'' basis} \qquad (24)$$

$$\text{Nitrogen} \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{nitrogen referred to ``moisture- and ash-free'' basis} \qquad (25)$$

$$(\text{Oxygen} - \tfrac{8}{9}\ \text{moisture}) \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{oxygen referred to ``moisture- and ash-free'' basis} \qquad (26)$$

$$\text{Calories} \times \frac{100}{100 - (\text{moisture} + \text{ash})} = \text{calories referred to ``moisture- and ash-free'' basis} \qquad (27)$$

NOTE.—Calorific values expressed in calories are converted to calorific values expressed in British thermal units by multiplying the former by 1.8.

The industrial demand for a laboratory test which would give

[illegible] compounds in the coal, or in the
[illegible] which compounds might affect the
[illegible] esponsible for the so-called "proxi-
[illegible] mate analysis these compounds are
[illegible] moisture; second, mineral impurities
[illegible] ganic or combustible matter, approxi-
[illegible] volatile matter and fixed carbon.
[illegible] the composition of a coal is expressed
[illegible] lphur, carbon, hydrogen, nitrogen, and
[illegible] constituents being equal to 100 per cent.

[illegible]ETHOD OF ANALYSIS

[illegible] moisture in coal consists of extraneous mois-
[illegible] xternal sources; and of inherent moisture, the
[illegible] he products of the original vegetable matter
[illegible] was derived.

[illegible] gram sample of the 60-mesh coal is placed in
[illegible] by $1\frac{3}{4}$-in. porcelain capsule, Fig. 3, the
[illegible] weight of the latter being known, and then
[illegible] for 1 hour at a temperature of 105°C.
[illegible] in the constant-temperature oven, shown
[illegible] after which it is removed from the oven,
[illegible] aluminum cover, Fig. 3, and cooled over sul-
[illegible] desiccator. The loss in weight multiplied by
[illegible] the percentage of moisture.

[illegible] oven, Figs. 4 and 5, embraces a double-
[illegible] double wall,
[illegible] the space
[illegible] with a solu-
[illegible] adjusted that
[illegible] at 15°C.
[illegible] 221°F.)
[illegible] the oven
[illegible] the
[illegible] the
[illegible] of air.
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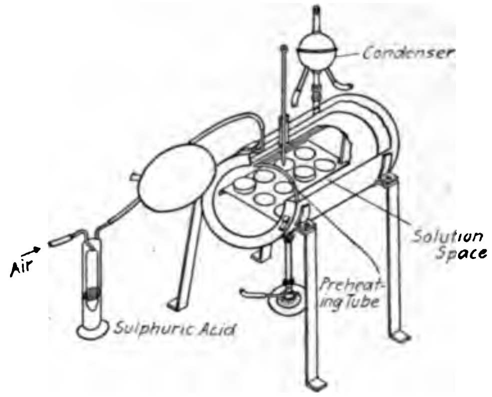

FIG. 4.—Constant-temperature oven.

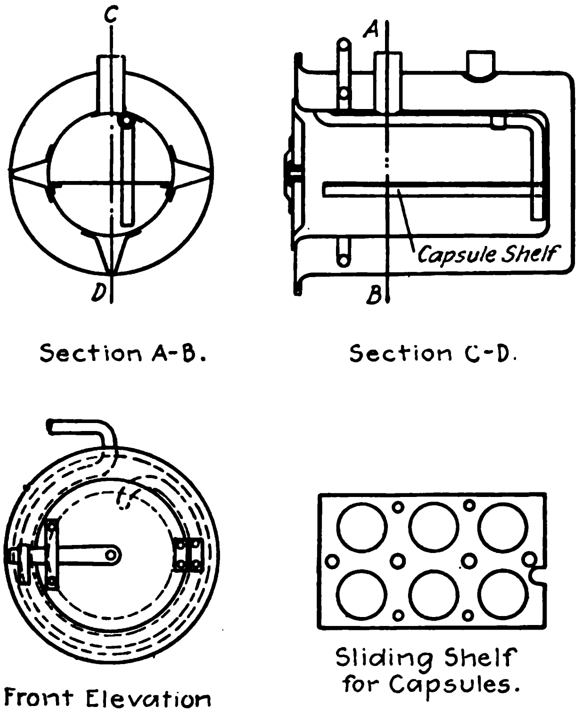

FIG. 5.—Constant-temperature oven.

the samples; this dried and preheated air absorbs the moisture from the samples, after which it is caused to escape through a small opening in the door of the oven. This air is forced through the oven at a speed regulated to change the total volume of air in the oven from two to four times per minute, and no trouble is experienced in maintaining the required temperature of 105°C. (221°F.) with this arrangement.

Ash.—Ash is an incombustible residue which is left after the complete combustion of the carbon contained in the coal; it is due to the inorganic matter in the coal, and is very largely composed of compounds of silica, alumina, lime, and iron, as well as smaller quantities of magnesia, titanium, and alkali compounds.

Ash is determined on the residue of dried coal from the moisture determination; this residue, in its porcelain capsule, is placed in a muffle furnace where it is slowly heated until the volatile matter is driven from the coal, the furnace being cool at the beginning of the determination. The object of this slow heating is to avoid coking the sample and thus making its burning difficult; also, if a coal high in volatile matter is subjected to rapid heating, the gas thus generated will have a tendency to explode within the capsule, thus mechanically carrying off portions of the ash. The ignition in the furnace is continued at a temperature of about 750°C. (1382°F.), the ash being stirred occasionally until all particles of carbon have disappeared, after which the capsule with its contents is removed from the furnace, cooled in a desiccator, weighed, and then replaced in the furnace, where it is again heated for half an hour, again cooled in the desiccator, and weighed. If the change in weight between these two weighings is less than 0.0005 gram, the weight is considered as constant and the weight of the capsule is deducted from the last weighing, but if the change in weight is greater then 0.0005 gram, the ash is again ignited and burned for ½ hour, cooled and weighed, this process being repeated until the change in weight between two successive ignitions is 0.0005 gram or less. The weight of capsule plus ash, minus the weight of the capsule, is then recorded as the weight of the ash. Some difficulty is often experienced in ignition to constant weight with coals containing a high percentage of iron, owing to the oxidation and reduction of iron oxides.

Ash as determined by this method represents the mineral matter remaining in the coal after ignition.

Volatile Matter.—Volatile matter and fixed carbon approximately represent the relative proportions of both the gaseous and solid combustible matters which are obtained by heating coal in a closed vessel, the principal constituents of the volatile matter being such combustible gases as hydrogen, carbon monoxide, methane, and other hydrocarbons as well as some non-combustible gases such as carbon dioxide and water vapor. Water which can be removed from the coal at a temperature of 105°C. (221°F.) is not included under volatile matter.

FIG. 6.—Platinum crucible.

A 1-gram sample of the 60-mesh coal is weighed into a bright, well-burnished, 10-gram (10-cc.) platinum crucible, provided with a close-fitting capsule cover, Fig. 6. This crucible and its contents is placed upon a platinum or nichrome triangle and heated to a temperature of 950°C. (1742°F.) for 7 minutes in the full flame of a No. 3 Meker burner, the gas orifice of this burner being adjusted to give a free burning flame from 16 to 18 cm. ($6\frac{1}{4}$ to 7 in.) high. The crucible should be so placed in the triangle that its bottom is 2 cm. ($\frac{3}{4}$ in.) above the top of the burner. It is necessary to protect the crucible from drafts, and a sheet iron chimney, constructed as shown in Fig. 7, is provided for this purpose.

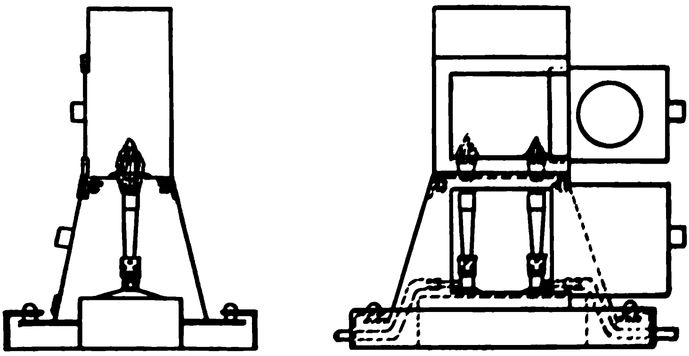

FIG. 7.—Crucible chimney.

The maximum temperature should be 950°C. (1742°F.) and this temperature is measured by a thermocouple whose hot junction is buried in the coal and in contact with the inside surface of the bottom of the crucible. After the coal has been heated for exactly 7 minutes, the crucible is cooled and weighed, the loss in weight minus the weight of moisture determined at 105°C. (221°F.) times 100 being the percentage of volatile matter.

Lignites and coals which are high in moisture content must be heated at a low temperature until all of the moisture has been expelled in order to avoid mechanical losses due to material which may be thrown out of the crucible by the rapid escape of steam and volatile matter.[1] This preliminary heating occupies from 4 to 6 minutes, after which the 7-minute heating in the full flame is applied.

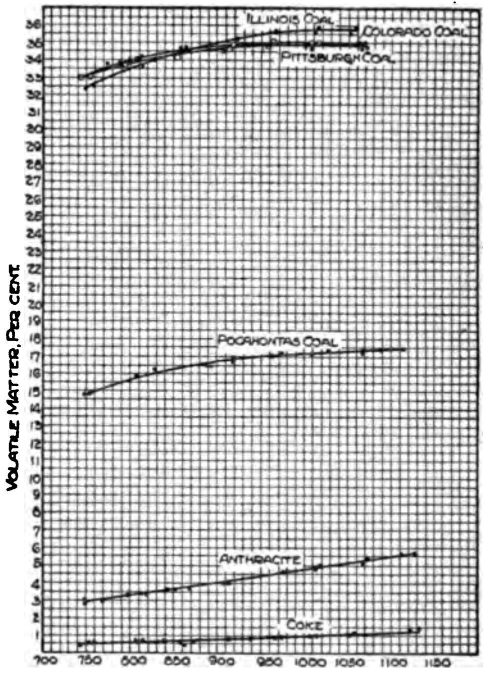

Diagram II.

It must be remembered that the volatile matter does not represent any definite compound in the coal; the above-described method of determination is entirely conventional, and any variation in the temperature or rate of heating will change the amount of volatile matter determined, this being graphically shown by the curves in Diagram II, which represent the amount of volatile matter obtained by heating a number of portions of a single sample of coal at various temperatures between 750°C. (1382°F.) and 1100°C. (2012°F.)[1]. This variation in temperatures with the consequent difference in volatile matter is quite likely to occur in such laboratories where different kinds of gas are used or where different gas pressures prevail, as shown by the experiments of FIELDNER and DAVIS.[2] Coal gas or water gas may be used to produce a crucible temperature of 950°C. (1742°F.) when supplied to the burner at a pressure of about 2 in. of water, but a change in pressure will cause a fluctuation in temperature. The use of natural gas requires a larger proportion of air for its complete combustion, and it should therefore be supplied at a pressure of not less than 10 in. of water to a good natural gas burner of the MEKER type, the burner being supplied with an ample quantity of air, otherwise the crucible temperature will be less than 950°C. (1742°F.). The use of the MEKER burner is recommended by the Bureau of Mines in place of the BUNSEN burner owing to its superiority over the latter, especially for natural gas. The construction of the MEKER burner practically eliminates the fluctuating inner cone of the ordinary BUNSEN flame, and thus produces a solid flame of fairly uniform temperature, this flame completely enveloping the bottom and sides of the crucible.

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[1] Bull. 28, Bureau of Mines, by N. W. Lord.

The maintenance of a crucible temperature of 950°C. (1742°F.) is the essential consideration in the determination of volatile matter; heating the crucible in a burner flame has the advantage of surrounding the crucible with flame, and practically avoiding the access of oxygen to the sample during determination, thus reducing the oxidation; but conversely, the flame temperature is difficult to control, as variations in the pressure and volume of the gas cause fluctuations in the temperature. If the gas pressure is subject to fluctuation, and a steady electric current is available, an electric furnace will afford the most uniform means of heating the sample, the additional oxidation due to the lack of flame protection being only about 0.1 per cent., provided a 10-cc. platinum crucible with a close-fitting capsule cover is used.[3]

The electric furnace used by the Bureau of Mines is of the vertical type, Fig. 8. Two of these furnaces are connected in series, controlled by one rheostat, a current of from 8 to 9 amp. being required to maintain the desired temperature of 950°C. (1742°F.) in each furnace, the drop in potential across the two furnaces under these conditions amounting to 60 volts. It has been found that the life of the heating element, under daily operation, is from 6 to 8 months. A platinum-platinum rhodium thermocouple is permanently installed in each furnace, each couple being connected to a millivoltmeter by means of a double-pole, double-throw switch. The position of the hot junction in each furnace must be so adjusted that the direct reading in the temperature scale of the millivoltmeter is identical with the maximum temperature produced inside the crucible, this condition being determined by means of another standardized couple the hot junction of which is in contact with the bottom of the interior of the crucible. When this comparison with the standardized couple is made, the usual platinum cover of the crucible is replaced

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[1] FIELDNER and HALL, Proc. 8th Intern. Cong. Appl. Chem., Vol. 10, 1912, page 139.

[2] FIELDER and DAVIS, Jour. Ind. Eng. Chem., Vol. 2, July, 1910, page 304.

[3] FIELDNER and HALL, Proc. 8th Intern. Cong. Appl. Chem., Vol. 10, 1912, page 148.

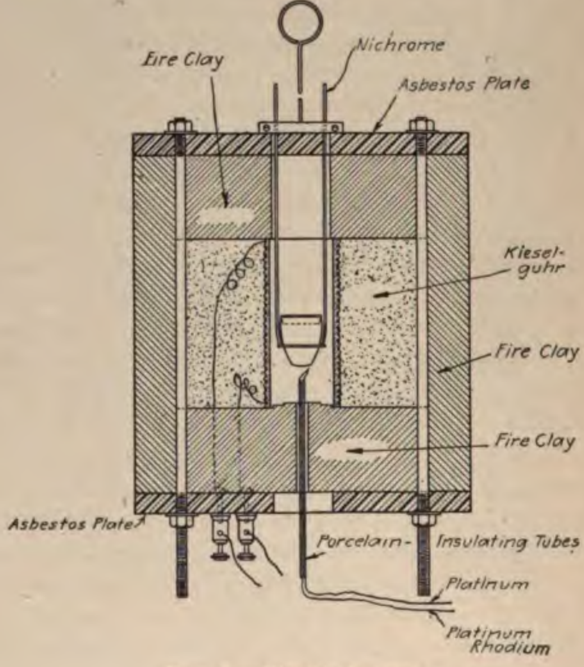

Fig. 8.—Electric furnace.

by one of nickel, the latter being perforated to admit the thermocouple leads, and to avoid any possible contamination of the thermocouple 1 gram of ignited silica or alumina is substituted for the coal in the crucible.

It seems almost impossible to construct two furnaces exactly alike, for it is usually found one runs from 20° to 60°C. (68° to 140°F.) hotter than the other when they are connected in series, and it is therefore necessary to equalize the temperatures by shunting a part of the current around the hotter furnace, a lamp-bank resistance being used for this purpose.

It is stated that adjustments such as this need be made only when setting up a pair of new furnaces, as there is practically no further change during the life of the furnaces.

Fixed Carbon.—Fixed carbon, which is not a pure carbon, is that residue of coke left in the crucible after the ash has been deducted, but it does not represent the total carbon in the coal. A large amount of the carbon is expelled in the form of volatile matter when it combines with hydrogen to form hydrocarbon, and with oxygen to form carbon monoxide and carbon dioxide.

The determination of fixed carbon is done by calculation, it being the result obtained by subtracting the sum of the percentages of moisture, ash, and volatile matter from 100.

Sulphur.—Sulphur is present in coal as a pyrite or marcasite, as a sulphate of iron, lime, and alumina, and also in combination with coal substances as organic compounds. In a proximate analysis of coal, sulphur is included in the volatile matter, fixed carbon, and ash, but if the ash contains no lime or alkali oxides, all of the sulphur is distributed between the volatile matter and the fixed carbon, approximately one-half the sulphur escaping with the volatile matter, the remainder being retained by the fixed carbon. Sulphur is one of the undesirable impurities in coal, although it generally exists in the form of combustible and thus contributes from 2250 to 2900 calories per gram of sulphur to the calorific value of the coal.

The method used for the determination of sulphur is known as the ESCHKA method, this "ESCHKA mixture" being made by thoroughly mixing two parts of light calcined magnesium oxide (MgO), with one part of anhydrous sodium carbonate (Na_2CO_3). A sample of 60-mesh coal, weighing 1.3736 grams, is placed in a 30-cc. platinum or porcelain crucible, where it is thoroughly mixed with 2 grams of the Eschka mixture, about 1 gram of the Eschka mixture being spread over the top to form a cover. The crucible containing this prepared sample is now placed in a slanting position on a triangle, the mixture being then burned out over an alcohol, gasoline, or natural-gas flame. Artificial gas is not recommended for this purpose, as it usually contains sulphur in an amount which may introduce an error in the determination due to the uncertainty as regards the quantity of sulphur which may be taken up by the mixture. A very low flame must be used in starting the burning in order to avoid driving off the volatile matter so rapidly that the sulphur may escape unburned; this condition can readily be detected, as even a small loss of sulphur dioxide (SO_2) will give off a very pungent odor. Care should be taken never to heat the crucible to such a degree as will cause a blackening of the Eschka mixture cover.

After the crucible has been heated very slowly and cautiously for about 30 minutes, the heat is increased until the crucible becomes red hot, when the contents should be occasionally stirred until all of the black particles in the mixture are burned out, this latter condition indicating that the process is complete. After the crucible and its contents have been allowed to cool, the contents are transferred to a 200-cc. beaker where they are digested for at least 30 minutes with 75 cc. of hot water, the contents of the beaker being then filtered into a 300-cc. beaker, the insoluble residue being washed twice with hot water by decantation, and then transferred to a filter paper where it is washed with small quantities of hot water until the volume of solution in the 300-cc. beaker amounts to about 200 cc. About 4 cc. of saturated bromine water (or a slight excess), together with sufficient concentrated hydrochloric acid (HCl) to make the solution slightly acid, are now added, after which the solution is boiled, and the sulphur is precipitated in the form of barium sulphate ($BaSO_4$) by adding 20 cc. of a hot 5 per cent. solution of barium chloride ($BaCl_2$), this solution containing the precipitate being allowed to stand for at least 2 hours at a temperature just below boiling. The following day this solution is filtered through an ashless filter paper and washed with pure hot water until a silver nitrate solution fails to show a precipitate with a drop of the filtrate. If it is desired to test for an excess of barium chloride, this may be done by adding a few drops of sulphuric acid solution to the filtrate. The precipitate is now placed in a weighed porcelain crucible and ignited, a free access of air being allowed, the paper being folded loosely over the precipitate in order to prevent spattering. Only sufficient heat should be applied at first to smoke the paper off very gradually, and the final heating should not exceed a dull red; after the paper has been completely burned off, the heating is continued for a few minutes, and the crucible is then cooled and weighed, the weight in grams of barium sulphate, after deducting the weight of barium sulphate found in a blank determination in which the same reagents are used, multiplied by 10, being equal to the percentage of sulphur in the sample.

When a large number of determinations are to be made simultaneously, the Bureau recommends the use of a muffle furnace heated electrically, by natural gas, or gasoline gas, as more convenient and rapid than heating the samples over individual gas or alcohol burners. In this case the crucibles containing the coal sample and ESCHKA mixture are placed in the cold muffle, the latter being then gradually heated until a temperature of about 850°C. (1562°F.) is reached, the total time consumed in this heating being usually from 1½ to 2 hours. After the crucibles are removed from the muffle the contents should be thoroughly stirred, and if any black particles remain in the crucibles they should be returned to the muffle and heated until all of the black particles are burned.

Carbon and Hydrogen.—In the determination of carbon and hydrogen two methods are used, the one with a gas-fired combustion furnace, and the other with an electrically heated combustion furnace.

Gas-Fired Combustion Furnace.—The carbon and hydrogen determinations are made with a weighed quantity of the sample in a 25-burner combustion furnace of the GLASER type, the products of combustion being thoroughly oxidized by being passed over red-hot copper oxide and lead chromate, being then fixed by absorbing the water in a weighed MARCHAND tube filled with granular calcium chloride ($CaCl_2$) and by absorbing the carbon dioxide in a LIEBIG bulb containing a 30 per cent. solution of potassium hydroxide (KOH).

The apparatus employed in this determination consists of a purifying train, arranged in duplicate, a combustion tube located within the furnace, and an absorption train. The purifying train consists of the following purifying reagents: sulphuric acid, potassium hydroxide solution, soda lime, and granular calcium chloride, these reagents being arranged in order of the passage of air and oxygen through them, one of the trains being for air, and the other for oxygen. The air and the oxygen are made to bubble through about 5 mm. of the purifying agent in the scrubbing bottles containing the sulphuric acid and the potassium hydroxide; both purifying trains are connected to the combustion tube by means of a "Y" tube, the joint being made tight by a rubber stopper.

The combustion tube is made of hard Jena glass, having an external diameter of about 21 mm., and a total length of 1 meter. The first 30 cm. of the tube is kept empty, an asbestos plug, acid washed and ignited, following this empty space, or a copper gauze roll may be used in place of the asbestos plug; the following 40 cm. of the tube is filled loosely with copper-oxide wire, followed by another asbestos plug which separates the wire from 10 cm. of fused lead chromate, the latter being held in place by a third asbestos plug located 20 cm. from the end of the tube. The end of the combustion tube is drawn out in order to facilitate the rubber tubing connection with the absorption trains.

The absorption train is made up, first, of a MARCHAND tube filled with granular calcium chloride ($CaCl_2$) for the absorption of moisture, this tube being followed by a LIEBIG bulb containing a 30 per cent. solution of potassium hydroxide (KOH), in which any possible impurities, such as ferrous iron or nitrates, have been oxidized by means of a small quantity of potassium permanganate ($KMnO_4$). A guard tube, containing soda lime and granular calcium chloride, is attached to the LIEBIG bulb in order to absorb any carbon dioxide which may escape the potassium hydroxide solution and any water which may be evaporated from that solution.

The train is now connected to an aspirator for the purpose of drawing the products of combustion through the entire train, a guard tube containing calcium chloride preventing moisture from running back into the absorption train, the suction being maintained constant by means of a MARIOTTE flask. It is stated that the advantage of aspirating the gases through the train, rather than forcing them through by pressure, is that the pressure on the rubber connections is from the outside, thus maintaining the gas-tight connections more easily than if the pressure were on the inside of the tube, as all connections should be as tight as possible. The test for tightness is usually made by starting the aspiration at the rate of about three bubbles of air per second through the potash bulb, after which the inlet for air and oxygen is closed at the opposite end of the train; the apparatus is considered tight if no more than three bubbles per minute appear in the potash bulb.

If the train has been idle for some hours, or if any changes n chemicals or connections have been made, a blank is run by aspirating about 1 liter of air through the train before starting a determination, the train being heated for this blank run in the same manner as if a determination on coal were being made; if the LIEBIG bulb and the tube containing calcium chloride show a change in weight of less than 0.5 mg. each, the apparatus is considered in proper condition for use.

A 2-gram sample of the 60-mesh coal is weighed into a platinum or porcelain boat, both the boat and its contents being placed in a glass weighing tube which is closed with a stopper to avoid moisture changes. After the absorption tubes are connected, the boat with its contents is transferred from the weighing tube to the combustion tube, it being essential that the latter be cool for the first 30 cm., while the copper oxide should be red hot and the lead chromate at a dull-red heat, the transfer of the boat from the weighing tube to the combustion tube being accomplished with the least possible delay. As soon as the boat has been placed in position near the asbestos plug which precedes the copper oxide, the stopper connecting with the purifying train should be inserted and the aspiration started with pure oxygen gas at the rate of three bubbles per second. One burner is turned on about 10 cm. back from the boat, aspiration being carefully continued until practically all of the moisture has been expelled from the sample, after which the heat is very gradually increased until all volatile matter has been driven off. It is essential that the heat be applied gradually while driving off the volatile matter, in order that a too rapid evolution of gas and tar may be avoided, as a rapid evolution may cause incomplete combustion of these substances, or they may be driven back into the purifying train. The gradual increase in heat is accomplished by turning on more burners under the open part of the tube, this being continued until the coal sample is ignited, after which the temperature can be increased rapidly, care being taken not to melt the combustion tube. This aspiration with oxygen is continued for 2 minutes after the sample ceases to glow, the heat being then turned off, followed by the aspiration of about 1200 cc. of air, after which the absorption bulbs are disconnected and weighed. The increase in weight of the calcium chloride tube multiplied by 55.55 will equal the percentage of hydrogen, while the increase in weight of the potassium bulbs multiplied by 136.36 will equal the percentage of carbon. The ash in the boat should now be weighed and carefully examined for any traces of unburned carbon, as if the latter should exist the determination would be valueless.

Electric Combustion Furnace.—The Heraeus type of electrically heated combustion furnace is also used for the determination of carbon and hydrogen, this furnace consisting of three independent heaters, (*a*), (*b*) and (*c*), Fig. 9; the heaters (*a*) and (*b*) are provided with wheels mounted on a track in order to permit of moving them along the tube, while the third heater (*c*) is stationary and surrounds the lead chromate; this latter heater is not supplied with the furnace by the manufacturers, but was devised and made in the Bureau, consisting of a 12 cm. long alundum tube wound with No. 16 nichrome II wire, the whole being enclosed in a cylinder packed with magnesia asbestos. The movable heaters have about 9 grams of thin platinum foil wound on a 30-mm. internal diameter porcelain tube, the larger one (*b*), which is used to heat the copper oxide, being 350 mm. long, and the smaller one (*a*), used to heat the 60-mesh sample in the boat, being 200 mm. long. The combustion tube (*d*), of Jena glass or fused silica, has an external diameter of about 21 mm. and a length of 900 mm., being supported in an asbestos-lined nickel trough (*e*). Separate rheostats (*f*) and (*g*), mounted on the frame of the furnace, are used for independently regulating the current through each heater, the two platinum wound heaters, (*a*) and (*b*), requiring an average current of about 4.5 amp. at 220 volts, but if rapid heating is required the amperage must be increased.

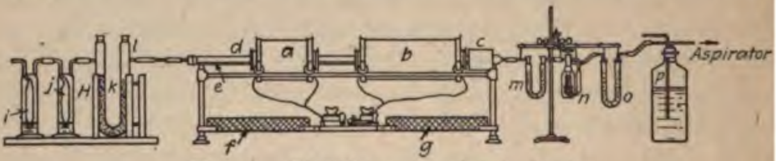

Fig. 9.—Electric combustion furnace.

A TAUBER's drying apparatus marked (*H*), Fig. 9, is used for purifying the oxygen or air before it enters the combustion tube, this apparatus containing the following reagents arranged in order of the passage of air or oxygen through them: sulphuric acid in (*i*), for removing any possible traces of ammonia; 30 per cent. potassium hydroxide (KOH) solution in (*j*); granular soda lime in (*k*), and granular calcium chloride in (*l*). A LINDE oxygen tank is directly connected on one side of the train, this tank being provided with a reducing valve for the purpose of regulating the oxygen pressure, the other side of the train being used for purifying the air supply.

The absorption train consists, first, of a 5-in. U tube (*m*), which is filled with granular calcium chloride ($CaCl_2$) for the absorption of moisture. Before this tube is placed in action, the calcium chloride is saturated with carbon dioxide in order to avoid the possible absorption of carbon dioxide during a determination by any traces of calcium oxide that may be present. This saturation is most conveniently accomplished by placing a quantity of calcium chloride in a large drying jar, which is then filled with carbon dioxide, the jar being allowed to stand over night, after which dry air is drawn through the jar to absorb the excess of carbon dioxide, the calcium chloride thus treated being kept in well-stoppered bottles for use.

The second element consists of a VANIER potash bulb (*n*), which is connected to the calcium chloride tube, the potash bulb containing a 30 per cent. potassium hydroxide solution and granular calcium chloride, this filling being good for six to eight determinations. The next element in the train consists of a guard tube (*o*), containing granular calcium chloride and soda lime, followed by a MARIOTTE flask (*p*), which in turn connects to the aspirator, the MARIOTTE flask being for the purpose of maintaining a constant pressure.

In general, the method of determination is identical with that explained for the gas furnace. The air can be warmed sufficiently to thoroughly dry the tube and its contents by moving the heaters toward the end of the tube where the gases enter, and by cutting in the electric current. The current is now cut off the small heater (*a*), and the large heater (*b*) is moved over the copper oxide; about 250 mm. of that portion of the combustion tube situated between the two heaters, or that portion of the tube where the boat containing the coal sample is to be placed, is kept exposed. The full current is then turned on the large heater and the copper oxide is brought to a red heat, the temperature being reduced when the copper oxide is red hot, by means of the rheostat, in order to avoid melting the tube. While this portion of the apparatus is being prepared the absorption train is weighed and connected, the boat containing the 60-mesh sample being placed in the exposed and cooler portion of the tube located between the two heaters.

The current is now passed through the shorter heater (*a*), and by a careful manipulation of the rheostat, and by gradually advancing the heater toward the boat, the rate of evaporation of moisture as well as the evolution of the volatile matter can be readily controlled.

After combustion has been completed, the current is cut off the smaller heater (*a*) and it is moved back to permit the tube to cool and thus prepare it for the next determination, the final aspiration of the air and the weighing of the absorption train being conducted in a manner identical to that given for the gas-furnace.

The term "available hydrogen" is based on the assumption that all of the oxygen contained in the coal is combined with the hydrogen constituent to form water. The amount of hydrogen which combines in this manner, and which is thus available for producing heat, is equal to one-eighth of the oxygen, the remainder of the hydrogen present, or the "available hydrogen," combining with the carbon and thus contributing to the calorific value of the coal.

Nitrogen.—The KJELDAHL-GUNNING method is used for nitrogen determination. One gram of the 60-mesh coal sample is boiled or digested with 30 cc. of concentrated sulphuric acid (H_2SO_4), 5 grams of potassium sulphate (K_2SO_4), and 0.6 gram of mercury in a 500-cc. KJELDAHL flask until all particles of the coal are oxidized and the solution becomes nearly colorless. This boiling should be continued for at least 2 hours after the solution has reached the straw-color stage, and the addition of a few crystals of potassium permanganate ($KMnO_4$), after digesting a few hours, will assist in reducing the boiling period required for the oxidation of anthracite. After the solution has been allowed to cool, it is diluted to about 200 cc. with cold water, after which 25 cc. of potassium sulphide (K_2S) solution (40 grams of K_2S per liter) is added in order to precipitate the mercury, and from 1 to 2 grams of granular zinc to prevent lumping. Sufficient saturated sodium hydroxide (NaOH) solution (from 80 to 100 cc.) to make the solution distinctly alkaline is gradually added, and the flask is immediately connected with a condenser.

As the loss of ammonia is possible, this danger may be reduced by holding the flask in an inclined position while the sodium hydroxide solution is being added, the alkaline solution running down the side of the flask and forming a layer below the lighter acid solution. The solution should be mixed by gently shaking the flask after it is connected to the condenser.

The ammonia (NH_3) is distilled over into 10 cc. of standard sulphuric acid solution, the latter carrying sufficient cochineal to act as an indicator for titration. The solution is distilled until about 200 cc. of distillate has passed over, the distillate being titrated with standard ammonia (NH_4OH) solution, 20 cc. of the NH_4OH solution being equal to 10 cc. H_2SO_4 solution, which in turn is equal to 0.05 gram of nitrogen.

The use of the 500-cc. KJELDAHL flask does not, however, necessitate the transfer of the solution to a distillation flask, as the KJELDAHL digestion flask can be connected directly with the condensing apparatus.

The Bureau makes routine determinations in sets of twelve, the digesting and distilling apparatus being designed for twelve flasks, one flask in each set containing a blank determination in which 1 gram of pure cane sugar (sucrose) is substituted in place of the coal sample. The nitrogen found in this blank determination is deducted from that found in each of the eleven coal determinations that were made in the same set. The sodium hydroxide and the potassium sulphide are usually dissolved in a single stock solution, a 25-lb. drum of electrolytic caustic soda being dissolved in 18 liters of water contained in a 6-gal. stoneware jar, potassium sulphide being then added until the quantity of alkali solution required for a nitrogen determination (from 85 to 100 cc.) contains 1 gram of potassium sulphide. As caustic soda 99 per cent. pure is used, the above proportions require 280 grams of potassium sulphide.

Phosphorus.—A 6.52-gram coal sample is burned to ash in a muffle furnace for a phosphorus determination, this ash being then mixed with from four to six times its weight of sodium carbonate plus 0.2 gram of sodium nitrate, after which it is fused at the highest temperature of the blast lamp. This fused mass is next dissolved in water, acidified, and thén evaporated to dryness, the residue being taken up in hydrochloric acid, while the phosphorus is determined either by weighing or by titrating the yellow precipitate with permanganate.

Calorific Value.—The calorific value of a fuel is defined as the total quantity of heat developed by the complete combustion of a unit weight of the fuel, the heat units being expressed in the metric system as the gram calorie (cal.), which is the quantity of heat required to raise the temperature of 1 gram of water 1°C. at 15°C., and the kilogram calorie (cal.), or the quantity of heat required to raise the temperature of 1 kilogram of water 1°C. at 15°C. In the English system this heat unit is known as the "British thermal unit," expressed as B.t.u., and it is the quantity of heat required to raise the temperature of 1 lb. of water 1°F. at 60°F., the metric calorie being converted into the British thermal unit by multiplying the former by 1.8.

Before the day of the bomb calorimeter, the heating value of a coal was usually determined from the ultimate analyses by one of the following equations of Dulong:

1. Calorific value in calories per gram =

$$8080C + 34{,}460\left(H - \frac{0}{8}\right) + 2250S \qquad (28)$$

2. Calorific value in B.t.u. per pound =

$$14{,}544C + 62{,}028\left(H - \frac{0}{8}\right) + 4050S \qquad (29)$$

In these equations C = carbon, H = hydrogen, O = oxygen, and S = sulphur.

The most accurate method of determining the calorific value is, however, accomplished by combustion in a bomb calorimeter, the method practised by the Bureau of Mines being applicable to either the Mahler, Atwater, Williams, or Emerson apparatus. Before being used, this instrument should be carefully standardized by burning standard samples of cane sugar, naphthalene, and benzoic acid, these substances, which can be obtained from the Bureau of Standards, having known calorific values. The standardization of the bomb should be conducted under exactly the same conditions and with the same thermometer, a calibrated thermometer being essential, as is used in the determinations.

In making a calorific determination, a 1-gram sample of the

60-mesh coal is placed in a platinum tray, the bottom of the tray being covered with a piece of asbestos paper that has been washed and ignited, the use of which avoids trouble in completely burning anthracite coal, one of the platinum terminals of the firing circuit being attached to the tray. The terminals are connected by a piece of iron wire 105 mm. long, 0.16 mm. in diameter, and weighing about 13 mg., or, in the case of a platinum-lined bomb, by means of platinum wire, the ends of the wire being attached to the *clean* platinum terminals by simply wrapping the wire around them, the iron wire being bent down sufficiently for it to touch the sample of coal on the mat within the tray. The tray is now placed in the bomb and the lid is screwed down tightly against the lead gasket. Oxygen is next forced into the bomb until the manometer records a pressure of from 18 to 20 atmospheres within the bomb, after which the needle-point valve is closed tight enough to prevent a gas leakage. The oxygen must be admitted very slowly in order to avoid blowing any coal dust particles out of the tray.

The bomb, after it has been filled with oxygen, is placed in the brass bucket of the apparatus, this bucket containing from 2000 to 2500 grams of distilled water, the amount of water used depending upon the type of calorimeter, the bucket having been previously placed in the insulated jacket, the temperature of this water being from 2.5° to 3°C. below the temperature of the water jacket. The stirring apparatus should be so adjusted that it does not touch either the bucket or the bomb, and the thermometer should be so inserted that the bulb stands about 5 cm. from the bottom of the bucket, care being taken that it does not come into contact with any metal portions of the apparatus, the outside terminals of the bomb being connected with wires leading to the switch. After the stirrer has been in operation for about 1 minute, or until the water has been thoroughly mixed, a first reading of the thermometer is taken, a reading telescope attached to a cathetometer being used for this purpose. The usual thermometer is graduated to 0.01°C., and the readings can therefore be interpolated to 0.001°C., but for accurate work the thermometer should be calibrated to 0.001°C. Care should be taken to stir the water so that it remains uniform during the determination, but the rate of stirring should be such as to insure a thorough mixing of the water. If covered calorimeters are used, the temperature should never be permitted to rise more than 1°C. above the temperature of the water jacket.

The reading telescope and the cathetometer, shown in Fig. 10, and which was devised by the Bureau, greatly improve the accuracy of the thermometer readings. By referring to the figure, it will be seen that the telescope is mounted on a sliding carriage (*a*), this carriage moving freely on the post (*b*) and being counterweighted by the weight (*c*) inside the post. The telescope used by the Bureau has a magnifying power of 55 diameters, and it may be adjusted at any angle, but once adjusted, this angle must be maintained throughout a determination, and rotation of the telescope around the post is prevented by a pin which projects from the carriage and slides in a groove in the brass post.

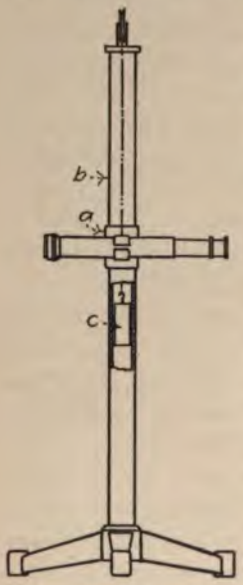

FIG. 10.—Reading telescope and cathetometer.

An actual determination may be divided into three periods: first, the preliminary period, which usually requires five readings, taken 1 minute apart, or until the rate of change per minute is nearly constant, the current at 20 volts being turned on for about ½ second after the fifth reading; this is now the beginning of the second or combustion period, during which the first two readings are taken at ½-minute intervals, owing to the great change in temperature, the temperature rising to a maximum and then falling; after the rate of fall has become uniform the readings are taken every minute for 5 or 6 minutes, the final reading of the combustion period being the first reading after the rate of fall becomes uniform, this stage being the final period.

The following figures are from an actual determination and show the method of calculating the result as well as the corrections applied (see Technical Paper 8, Bureau of Mines, STANTON and FIELDNER):

METHOD OF CALCULATING FROM CALORIMETER READINGS

(Sample No. 10743. Weight 1.0000 gram.)

Time, P.M.	Reading, °C.			
1.00				
1.54	23.874		0.0058 rate of change in preliminary period.	26.463
1.55	23.897			23.897
1.56	23.885		Observed temperature change	2.566
1.57	23.892		Thermometer correction	0.002
			Stem correction	0.000
1.58	23.897	+ 0.0058[1]		2.564
		+ 0.0027[2]	Radiation loss	0.0066
				2.5706
1.58½	24.160	+ 0.0049[1]	Water equivalent	0.3000
		+ 0.0014[2]		
1.59	25.430	+ 0.0008[1]	Total heat developed (calories)	7,711.8
		− 0.0006[2]	Correction	40.3
1.60	26.280	− 0.0020[1]	Heat developed by combustion	
		− 0.0023[2]	of sample (calories)	7,671.5
2.01	26.439	− 0.0025[1]		
		− 0.0026[2]		
2.02	26.463	− 0.0026[1]		
		− 0.0026[2]		
2.03	26.466	− 0.0026[1]		
		− 0.0026[2]		
		− 0.0066 algebraic sum.		
2.04	26.463			
2.05	26.460			
2.06	26.456	− 0.0026 rate of change in final period.		
2.07	26.455			
2.08	26.454			
2.09	26.450			

	Calories
Wire burned = 11.2 mg.	= 17.9
Titer (1 cc. = 5 cal.), 2.5 cc.	= 12.5
Sulphur (0.01 mg. = 13 cal.), 0.0076 mg.	= 9.9
Room temperature = 24°C.	40.3

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[1] Computed rate per minute of temperature change at each reading.

[2] Temperature correction for heat loss during each interval.

The temperature reading of 23.874°C., taken at 1.54 P.M., is the first reading of the preliminary period, that of 23.897°C. taken at 1.58 P.M., or 4 minutes after the first, being the last reading of this period; therefore the total temperature change during this first period amounts to 23.897° − 23.874° = 0.023°, and the rate of change per minute during this period is 0.023° ÷ 4 = 0.0058°. In like manner, the rate of change per minute during the final period is found to be $\frac{26.463° - 26.450°}{5} = 0.0026.°$

The rate of change during the period of combustion is + 0.0058° (− 0.0026°) = 0.0084°, the observed change in temperature during this period being 2.564°, the change in rate per degree of temperature change during the combustion period therefore being $\frac{0.0084°}{2.564°} = 0.0033°$.

During the first half minute of the combustion period we find the change in temperature to be 0.263°, hence,

Change in rate during first half minute of combustion period =

$$0.263° \times 0.0033° = 0.0009°$$

rate of temperature change at 1.58½ P.M. =

$$0.0058° - 0.0009° = 0.0049°$$

and the rate of change for each succeeding reading during the combustion period is calculated in the same manner.

In order to calculate the temperature correction for the heat loss during the half-minute readings, the algebraic sum of the rates is divided by 4, and for the minute readings the algebraic sum is divided by 2 because the mean rate per minute of temperature change, multiplied by the time of the interval in minutes, is equal to the temperature correction for heat loss during each interval.

As an example, we find that at 1.58 P.M. the rate per minute of the temperature change was +0.0058°, and at 1.58½ P.M. this change was +0.0049°; therefore the mean rate per minute of temperature change for the ½-minute interval was

$$\frac{0.0058° + (+0.0049°)}{2} = +0.0054°$$

and ½ × (+0.0054°) = +0.0027°, this being the correction for heat loss which occurred during the half-minute interval.

The algebraic sum of all the temperature corrections for heat loss during each interval must be added to the observed temperature change after corrections have been made for the errors and stem correction in the thermometer, and this figure multiplied by the water equivalent of the apparatus will give the total heat developed; this result must be further corrected by deducting the value of the heat due to the formation of aqueous nitric acid, sulphuric acid, and to the combustion of the iron wire.

The correction for nitrogen burned to nitric acid is found by titrating the acidity of the bomb liquor with a standard ammonia solution of 0.00587 gram NH_3 per cubic centimeter, which is equivalent to 5 calories per cubic centimeter.

The correction for sulphuric acid, or for the burning of sulphur to H_2SO_4, is equal to 13 calories per 0.01 gram of sulphur, and, finally, the correction for iron wire is equal to 1.6 calories per milligram.

Sulphur in the Washings from a Bomb Calorimeter.—After combustion in the calorimeter the latter is thoroughly washed out, the washings being collected in a 250-cc. beaker. These washings are then titrated with standard ammonia solution of 0.00587 gram NH_3 per cubic centimeter, methyl orange being used as an indicator, in order to obtain "acid corrections" for the heating value, the acidity being due to the nitric acid (HNO_3) formed from the nitrogen of the coal and that of the air within the bomb, as well as to the sulphuric acid (H_2SO_4) formed from the combustion of the sulphur in the coal, after which 5 cc. of dilute (1:2) hydrochloric acid is added and the solution heated to boiling. The insoluble matter is filtered off and washed five times with hot water, after which the filtrate and washings, which together should have a total volume of 200 cc., are heated to boiling, the sulphur being then precipitated with barium chloride with the result that

$$\frac{\text{Weight of } BaSO_4 \times 13.74}{\text{Weight of sample}} = \text{percentage of sulphur.}$$

The results obtained by this method are usually from 3 to 8 per cent. lower in total sulphur present than are those obtained by the Eschka method, this difference being due to the loss of sulphur trioxide in the gas which escapes from the bomb and to the retention of some sulphur in the coal ash, but it has an advantage over the ESCHKA in that it effects a material saving in time. The results obtained are, however, sufficiently accurate for use in determining the purchase of fuel on a B.t.u. basis.

In order to facilitate the determinations, the ammonia solution should be made of such strength that 1 cc. of the solution is equivalent to 0.00483 gram of nitrogen, because this weight of nitrogen when burned to nitrogen pentoxide (N_2O_5) plus water generates exactly 5 calories of heat, these figures being derived from the fact that the calorific value of nitrogen burned to N_2O_5 + water is equal to 1035 calories per gram; therefore

$$1035 \text{ calories} : 5 \text{ calories} :: 1 \text{ gram} : 0.00483 \text{ gram}$$

hence 0.00483 gram of nitrogen when burned to aqueous nitric acid will generate 5 calories of heat, these figures being derived as follows:

The standard ammonia solution is made up according to the equation

$$HNO_3 + NH_3 = NH_4NO_3 \tag{30}$$

and as the atomic weight of N is 14 and the molecular weight of NH_3 is 17, we have

$$14 : 17 :: 0.00483 \text{ gram} : 0.00587 \text{ gram}$$

therefore 0.00587 gram of NH_3 is equivalent to 0.00483 gram of nitrogen, and when this is burned to aqueous nitric acid (HNO_3) the heat generated amounts to $1035 \times 0.00483 = 4.99905$, or 5 calories; hence in preparing the solution, 5.87 grams of NH_3 are added to 1 liter of water, but this solution must not only satisfy the nitric acid formed, it must also satisfy the sulphuric acid resulting from the combustion of the sulphur in the coal, and the strength of the ammonia solution in terms of sulphur transformed into sulphuric acid (H_2SO_4) is determined from the following equation:

$$2NH_3 + H_2SO_4 = (NH_4)_2SO_4 \tag{31}$$

and $2NH_3 : S = 34 : 32 :: 0.00587 \text{ gram } NH_3 : 0.0055 \text{ gram } S$.

The heat of combustion when sulphur is burned to aqueous sulphuric acid amounts to 4450 calories per gram of sulphur, this reaction with its accompanying heat being produced in the bomb calorimeter when fuel is burned in an atmosphere of oxygen under high pressure, but in the ordinary combustion of coal, or in a boiler furnace, the sulphur is burned to sulphur dioxide (SO_2), which has a heat of formation equal to 2250 calories per gram of sulphur; the difference between these two values, or those due to the formation of aqueous sulphuric acid in the one case and to sulphur dioxide in the other, amounts to $4450 - 2250 = 2200$ calories per gram of sulphur.

This difference necessitates a correction of 2200 calories per gram of sulphur in the determination, because of the formation of aqueous sulphuric acid, and since 1 cc. of the standard ammonia solution is equivalent to 0.0055 gram of sulphur, we have $0.0055 \times 2200 = 12.1$ calories as the heat correction which must be made if all the acidity of the liquor from the bomb is represented in sulphuric acid.

As a general statement it may therefore be said that the ammonia solution containing 0.00587 gram (NH_3) per cubic centimeter is equivalent to 5 calories when nitrogen is burned to aqueous nitric acid, or to 12.1 calories when sulphur is burned to aqueous sulphuric acid, this difference therefore requiring a further correction for the sulphur that may be determined separately. This correction is found to be a function of the difference between the value of the ammonia solution in terms of sulphur, or 12.1 calories, and its value in terms of nitrogen, or 5 calories, the difference being $12.1 - 5 = 7.1$ calories; this difference is divided by 0.0055, the value of 1 cc. of the ammonia solution in grams of sulphur, and $\frac{7.1}{0.0055} = 1291$ calories per gram of sulphur, or practically 13 calories for each 0.01 gram of sulphur. We therefore find that the total correction for acidity is equal to the cubic centimeters of ammonia solution multiplied by 5 plus the centigrams of sulphur multiplied by 13, the figure 5 being the factor for nitric acid.

The MAHLER bomb calorimeter is shown in Fig. 11, where

A = the water jacket by means of which constant conditions are maintained outside of the calorimeter proper, thus making it possible to secure a more accurate computation of losses due to radiation.

B = the porcelain-lined steel bomb, which is filled with compressed oxygen, and in which the combustion of the coal takes place.

C = the platinum pan containing the coal.
D = the calorimeter proper, which surrounds the bomb and which contains a definite quantity of weighed water.
E = an electrode for igniting the coal contained in the pan *C*.
F = a fuse wire connected to the electrode *E*.
G = a support for the agitator or water mixer.
I = a thermometer for determining the temperature of the water in the calorimeter. This thermometer should be supported on a stand which is independent of the calorimeter, as if it is supported directly by the latter it is subject to some movement occasioned by possible tremors in portions of the calorimeter, and these tremors will make temperature readings difficult.

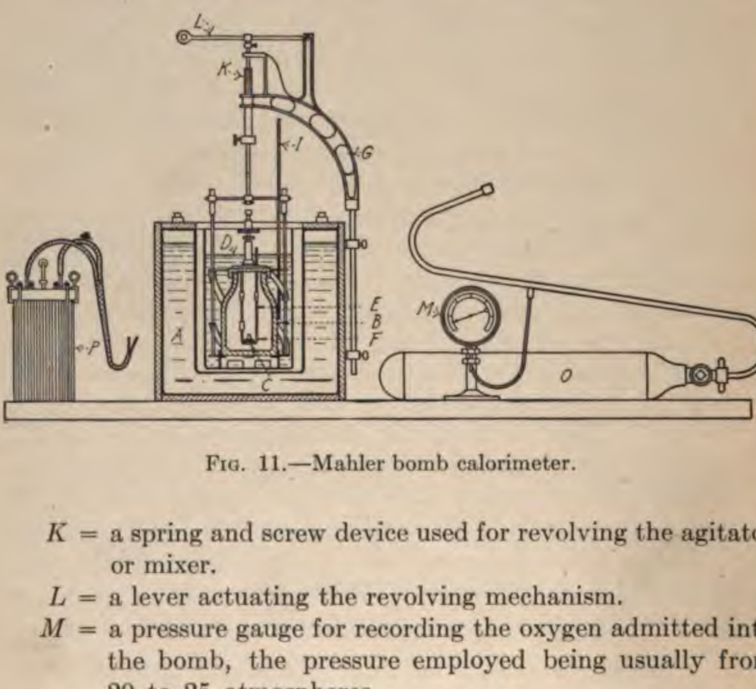

Fig. 11.—Mahler bomb calorimeter.

K = a spring and screw device used for revolving the agitator or mixer.
L = a lever actuating the revolving mechanism.
M = a pressure gauge for recording the oxygen admitted into the bomb, the pressure employed being usually from 20 to 25 atmospheres.
O = a tank containing the oxygen.
P = a series battery for supplying the current required to fuse the wire used in igniting the fuel. A lighting system current should not be used for this purpose, as results may be affected due to sparking while burning the fuse.

A constant temperature device, Fig. 12, used by the Bureau of Mines, provides a water supply at the proper temperature for starting calorimeter determinations; this device consists of a large copper tank (*A*) containing an agitator or mixer (*a*), the latter being revolved through the medium of a small electric motor, and an electric thermostat (*b*), which operates an electromagnetic valve (*c*) located in the ice water tank (*B*). After a calorimetric determination has been completed, the water, which has been slightly warmed in the bucket, is returned to the tank (*A*). The addition of this warmer water acts on the thermostat and thus causes the latter to close an electric circuit through a dry battery and relay, the latter in turn closing a 220-volt circuit through the coil of the solenoid (*d*), this closing of the circuit causing the valve (*c*) to open and thus admit ice water into the copper tank (*A*). The admission of this water lowers the temperature of the water in (*A*), and when the proper temperature has been thus produced, the circuit is automatically broken by the thermostat, and valve (*c*) is closed. The water required for each determination is drawn off from tank (*A*) through the cocks (*e*) and (*f*) to the calorimeter bucket, where it is weighed. A tare weight, equal to the total water equivalent of the calorimeter plus the weight of the empty bucket, is provided by means of a strong, glass, shot-filled bottle.

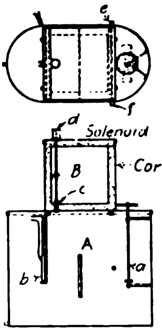

Fig. 12.—Constant temperature device.

Before being placed in use, the calorimeter should be standardized by determining the water-equivalent value of the instrument, this being done by the method of specific heats, the electrical method, the method of mixtures, the differential method, or by the method which employs the combustion of a substance having a known calorific value.

An example of standardization by the first or specific heat method, made on the Mahler bomb by the Bureau of Mines,[1] is given below; this bomb showed a water equivalent of 500 as determined from the average of fifteen combustions of pure naphthalene, benzoic acid, and sugar cane, a check on these determinations by the method of specific heats giving the following result:

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[1] Tech. Paper 8, Bureau of Mines, Stanton and Fieldner.

T = the change of temperature, corrected for radiation and thermometric errors.

H. H. Clark[1] has devised the following equation for determining the calorific value in British thermal units, this equation being the result of more than 500 analyses of coals the heating value of which had been determined by means of a standard calorimeter. Clark states that the results given by the equation are within 2 per cent. of those obtained by the calorimeter.

B.t.u.'s per pound of dry fuel =

$$100 \frac{(C.c. + V.v.) - (16M + 30A + 39S)}{100 - M} \qquad (33)$$

in which

C = per cent. of carbon in coal.
c = constant, as given below.
V = per cent. of volatile in coal.
v = constant in Table XXVI.
M = per cent. of moisture in coal.
A = per cent. of ash in coal.
S = per cent. of sulphur in coal.

For "c" Use

For anthracite coal .... $c = 141$
For bituminous coal .... $c = 140$
For lignite .... $c = 95$
For coke .... $c = 130$

Table XXVI

Per cent., vol.	"v"	Per cent., vol.	"v"	Per cent., vol.	"v"	Per cent., vol.	"v"
1	240.5	14	202.7	27	180.5	40	162.2
2	236.4	15	200.9	28	178.8	41	161.1
3	232.5	16	199.2	29	177.2	42	160.0
4	228.8	17	197.5	30	175.2	43	158.9
5	225.3	18	195.7	31	173.9	44	157.8
6	221.9	19	194.0	32	172.3	45	156.7
7	218.6	20	192.3	33	170.7	46	155.6
8	215.4	21	190.6	34	169.2	47	154.5
9	212.3	22	188.9	35	167.8	48	153.4
10	210.3	23	187.2	36	166.6	49	152.3
11	208.3	24	185.5	37	165.5	50	151.2
12	206.4	25	183.8	38	164.4	51	150.1
13	204.6	26	182.1	39	163.3	52	149.0

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[1] Amer. Gas Light Jnl., Sept. 30, 1912, page 217.

The calorific value of a coal, taken in conjunction with its proximate analysis, may give some idea as to the calorific value which will be found in the volatile products were it not for that portion of this volatile combustible which is transformed into tar, the latter having a high calorific value; therefore the calorific value of the coal can be accepted only as an index of what may be expected in the gas. It is a well-known fact that, as a rule, high volatile coals, having a high calorific value, are more apt to produce gas with high calorific values than those coals which have a low heating value. Carbonizing within working limits at low temperatures will produce more tar than when the same coal is carbonized at a higher temperature, and as a rule this low-temperature tar will be of a better quality and will contain less free carbon.

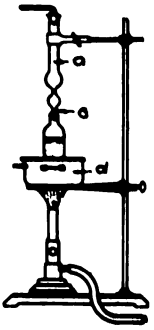

FIG. 13. Pycnometer for true specific gravity.

The True Specific Gravity.—In determining the *true* specific gravity of the coal, approximately 3.5 grams of the 60-mesh sample are weighed and placed in a 50-cc. pycnometer which contains about 30 cc. of distilled water. In order that no portion of the sample be lost during boiling, a one-bulb, 6-in. drying tube (*a*) is connected to the pycnometer by means of a small piece of pure rubber tubing (*c*), Fig. 13, the other end of the drying tube being connected to the aspirator. Suction is now applied and the contents of the flask are caused to boil gently under a partial vacuum for 3 hours, thus expelling all air from the coal sample; a water bath (*d*) is interposed between the flame of the burner and the pycnometer. After the air has been expelled, the pycnometer is detached, almost filled with boiled and cooled water, and then allowed to cool to the temperature of the room, being then stoppered and weighed, the temperature of the pycnometer contents being taken immediately after weighing. Each pycnometer must be accurately calibrated and a table should be constructed giving its individual capacity in grams of water at different temperatures. The true specific gravity is now determined by means of equation (34):

$$\text{True specific gravity} = \frac{W}{W - (W' - P)} \qquad (34)$$

in which

W = weight in grams of dry coal = weight in grams of the sample less its moisture content.
W' = weight in grams of the pycnometer plus the dry coal plus the filling water.
P = weight in grams of the pycnometer plus the filling water.

A Hogarth flask is also used to great advantage in determining the true specific gravity, and it is more convenient and accurate than the ordinary pycnometer; with the latter flask it is somewhat difficult to insert the stopper without catching some floating particles of the sample between the neck and the stopper, but this difficulty is removed by using the Hogarth flask.

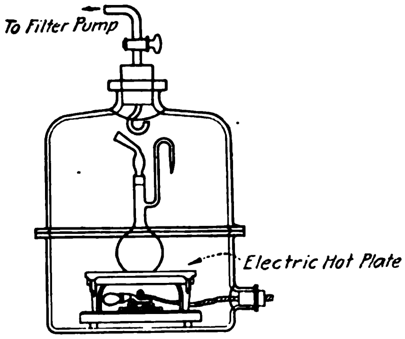

Fig. 14.—Vacuum desiccator for true specific gravity.

A 10-gram sample of the 60-mesh coal is weighed and carefully placed in the weighed Hogarth flask, the latter being half full of distilled water, the capacity of these flasks being from 100 to 125 cc. The flask is placed on a small electric hot-plate in a 10-in. vacuum desiccator, Fig. 14, this desiccator being evacuated by means of an aspirator or air pump, sufficient electric current being passed through the hot-plate to keep the water boiling; about 30 minutes are required to expel all of the air if an efficient vacuum pump is used. The flask is now removed from the desiccator, filled to the tubulure with distilled water, which has been recently boiled and cooled, and the stopper is inserted, a thin film of vaseline being applied to the stopper to prevent leakage.

The flask is now placed in a water thermostat and cooled to about 25°C., after which distilled water, which has been cooled in the same thermostat, is drawn through the tubulure until the water reaches a level slightly above the mark on the capillary of the stopper. By inserting the end of the tubulure in a small beaker containing the distilled water, and by applying slight suction on the stopper, the flask may be filled without removing it from the thermostat, and the flask must remain in the thermostat until the contents have reached a temperature of exactly 25°C. (77°F.). The water level is adjusted to the mark on the capillary by touching the end of the tubulure with a piece of filter paper and thus absorbing an excess, or by drawing in a small amount of additional water. The flask is now removed from the thermostat, wiped dry, weighed, and the true specific gravity determined by equation 34, the value of P being determined by filling the flask with boiled water, cooling, and weighing.

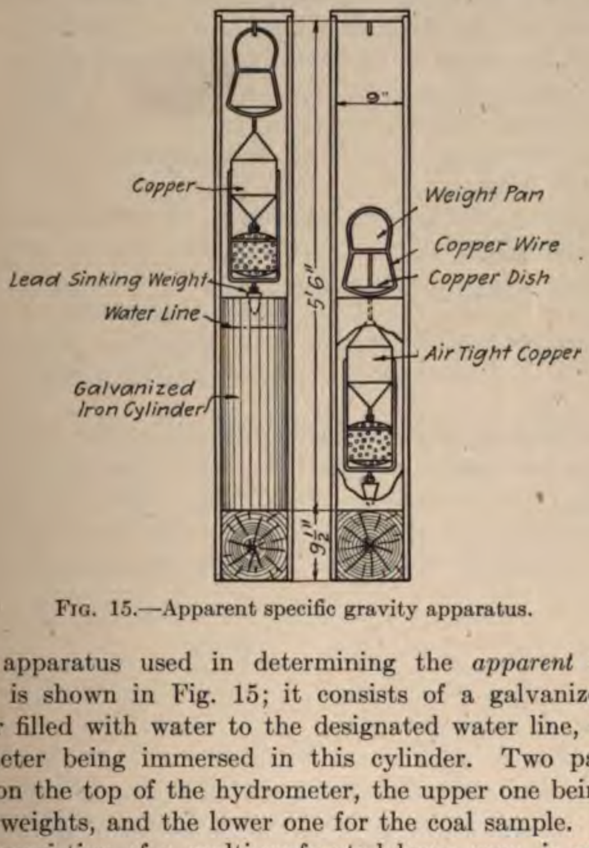

FIG. 15.—Apparent specific gravity apparatus.

The apparatus used in determining the *apparent* specific gravity is shown in Fig. 15; it consists of a galvanized iron cylinder filled with water to the designated water line, a brass hydrometer being immersed in this cylinder. Two pans are placed on the top of the hydrometer, the upper one being used for the weights, and the lower one for the coal sample. An air escape consisting of a multi-perforated brass cage, is provided beneath the air buoy to permit the air to escape when the instrument is immersed in the water, this cage carrying the sample when it is weighed under water.

In determining the *apparent* specific gravity, brass weights are placed on the upper pan and the hydrometer is allowed to sink until it reaches a mark on the stem between the copper pan and the buoy, this total weight being recorded. The weights are then removed and about 500 grams of the coal in 1½- to 2-in. cubes are placed in the copper dish, after which sufficient brass weights are applied to cause the hydrometer to sink to the mark on the stem, the difference in the weights used giving the weight of the sample in air. The sample is now removed and carefully placed in the brass cage below the buoy, the weights on the upper pan being adjusted until the instrument again sinks to the designated mark on the stem. Then the weight of the coal in water will be equal to the weight required to sink the hydrometer to the mark with no sample on the upper pan nor in the brass cage, less the weight required to sink it to the same mark with the sample immersed in the cage.

If the weight of the coal sample in air $= x$, and the weight of the same sample in water $= y$, the apparent specific gravity will be $\frac{x}{x - y}$. Therefore, $100 \times \frac{\text{apparent specific gravity}}{\text{true specific gravity}} =$ percentage by volume of coal substance, and $100 -$ percentage by volume of coal substance = percentage by volume of cell space.

Before making specific gravity determinations the samples should be thoroughly dried, and they should also be in lumps of approximately the same size and shape. When the sample is immersed, the hydrometer should be moved up and down in the water a number of times so that air bubbles may be expelled.

CHAPTER V

PREPARATION AND STORAGE OF COAL

Good coking coals are capable of producing a good, sound coke without any other preparation than crushing, but as many of the coals used for this purpose contain slate, iron pyrites, ferrous sulphide, sulphate of lime, argillaceous matter, and phosphorus, coal washing must be resorted to in order that the required quality of coke may result, the quality or cleanliness of the coal having a direct bearing on the yield of coke. Table XXVII[1] shows approximately the quantity of coal, in tons and pounds, required to produce a ton of coke in 1880, 1890, 1900, and annually since 1901, from which it is evident that up to 1903 the amount of coal required to produce a ton of coke exceeded 3100 lb.; from 1904 to 1910, inclusive, this weight was somewhat reduced, being between 3000 and 3100 lb., while for the last 3 years it was slightly less than 3000 lb., the minimum having been reached in 1912.

Table XXVII.—Coal Required to Produce a Ton of Coke

Year	Tons	Pounds	Year	Tons	Pounds
1880	1.57	3140	1906	1.531	3062
1890	1.56	3120	1907	1.519	3038
1900	1.57	3140	1908	1.515	3030
1901	1.57	3140	1909	1.510	3020
1902	1.56	3120	1910	1.513	3026
1903	1.56	3120	1911	1.499	2998
1904	1.544	3088	1912	1.491	2982
1905	1.537	3074	1913	1.496	2992

The coke yield since 1903 has increased in a ratio corresponding to the reduction in the amount of coal required to produce a ton of coke, and at present about 160 lb., or 5 per cent., less coal is required per ton of coke than was necessary 10 or 12 years ago, or before the retort coke ovens began to make their existence a factor in the coke-making industry.

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[1] The Mfg. of Coke in 1913, E. W. Parker, U. S. Geological Survey.

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According to Parker, the yield of coal in coke, to 1901, was usually less than 64 per cent., and only in 1 year did this yield reach 65 per cent.; however, it is doubtful if the coke yield ever exceeded 60 per cent. in the earlier years of the industry, as much of the coal used in those days was never accurately weighed, the amount coked being determined by estimating the cubical contents of the larrys used in charging the ovens, or by the number of mine cars sent to the coking plant.

Retort or by-product coke became an appreciable factor at the beginning of the present century, and its influence has increased each year with the growth of the proportion of retort coke to the total production, the increasing yield of coal in coke since 1901 bearing direct evidence of the influence of retort coke on the trade. According to Parker, the yield of coal in coke from 1902 to 1904, inclusive, was over 64 per cent.; during the next 3 years it was over 65 per cent., and since 1908 the yield has been 66 per cent. and more, reaching the maximum of 67.1 per cent. in 1912, but being reduced to 66.9 per cent. in 1913, this smaller yield being due to a smaller proportion of West Virginia coal and a larger proportion of Illinois or Indiana coal being coked at some of the retort oven plants in those states.

Table XXVIII shows the percentage yield of coal in coke in each state from 1909 to 1913, inclusive, and Table XXIX, the percentage of coal in coke in the United States from 1880 to 1913 inclusive.

The coal charged into the ovens in a number of the coking districts was the slack coal produced through mining, but the larger quantity used was run of mine, some of which was crushed in order to produce a more uniform quality of coke, while a considerable quantity of both slack and run of mine coal was washed in order to remove such impurities as slate, pyrite, etc., 19.8 per cent. of all the coal coked in coke ovens in 1913 being washed, while 80.2 per cent. was used without any other preparation than crushing, and the latter only in some instances.

In Pennsylvania and West Virginia, as well as in those by-product coke-producing states which receive their coal supplies chiefly from Pennsylvania and West Virginia, the greater portion of the coal charged to the coke ovens is not washed, most of the coal used in West Virginia being unwashed slack, while the unwashed coal used in the other states is for the greater part run of mine. Most of the coal used in Alabama is washed, nearly

TABLE XXVIII.—PERCENTAGE YIELD OF COAL IN COKE BY STATES[1]

State	1909	1910	1911	1912	1913
Alabama	60.7	61.6	62.6	64.9	63.6
Colorado	64.9	66.6	66.6	66.0	65.1
Georgia	53.8	54.8	51.7	50.0	51.5
Illinois	75.9	76.8	77.1	76.2	74.9
Indiana	44.4	78.3	80.6	81.8	77.1
Kansas	00.0	75.2	70.0	70.0	00.0
Kentucky	52.0	51.7	55.9	62.4	61.9
Maryland	67.9	65.6	66.2	65.8	69.4
Massachusetts	77.7	77.3	77.4	75.5	76.3
Michigan	74.1	75.7	74.2	75.4	76.2
Minnesota	67.7	68.0	66.7	69.6	66.9
Montana	44.7	44.7	00.0	00.0	00.0
New Jersey	77.7	76.1	76.2	78.4	75.4
New Mexico	53.9	61.6	61.5	60.9	59.4
New York	72.0	71.7	71.8	72.6	71.1
Ohio	65.4	68.3	68.2	69.2	69.3
Oklahoma	00.0	45.0	00.0	00.0	00.0
Pennsylvania	67.3	66.7	66.7	66.5	66.6
Tennessee	53.1	54.0	52.6	54.0	52.5
Utah	53.7	54.9	59.0	56.8	56.9
Virginia	65.4	64.6	63.9	62.2	64.7
Washington	61.7	63.0	66.6	62.6	64.2
West Virginia	62.0	61.1	60.4	60.7	61.3
Wisconsin	76.1	77.4	74.9	69.6	76.2
Total average	66.2	66.1	66.7	67.1	66.9

TABLE XXIX.—PERCENTAGE YIELD OF COAL IN COKE IN THE UNITED STATES[1]

Year	Per cent.	Year	Per cent.	Year	Per cent.	Year	Per cent.
1880	63.0	1902	64.1	1906	65.3	1910	66.1
1890	64.0	1903	64.1	1907	65.8	1911	66.7
1900	63.9	1904	64.8	1908	66.0	1912	67.1
1901	63.7	1905	65.1	1909	66.2	1913	66.9

two-thirds of the total being washed slack, while in Georgia, New Mexico, and Washington all of the coal is washed, the first two states using slack exclusively. In Colorado the greater portion of the coal coked is run of mine, washed, and in Virginia there is almost an equal division between slack and run of mine, but none of it is washed in this state.

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[1] The Mfg. of Coke in 1913, E. W. PARKER, U. S. Geological Survey.

In 1913 the total quantity of coal charged into ovens for the production of coke amounted to 69,239,190 short tons, of which 49,566,720 tons unwashed, and 4,793,053 tons washed were run of mine coal, while 5,958,173 tons unwashed, and 8,921,244 tons washed were slack coal, making the total amount of unwashed coal 55,524,893 tons as against 13,714,297 tons of washed coal. Table XXX shows the character of coal used in the production of coke from 1890 to 1913.

TABLE XXX.—CHARACTER OF COAL USED IN MAKING COKE. SHORT TONS[1]

Year	Run of mine		Slack		Total
	Unwashed	Washed	Unwashed	Washed	
1890	14,060,907	338,563	2,674,492	931,247	18,005,209
1895	15,609,875	237,468	3,052,246	1,948,734	20,848,323
1900	21,062,090	1,369,698	5,677,006	4,004,749	32,113,543
1905	31,783,314	3,187,994	8,196,226	6,363,143	49,530,677
1910	42,554,324	5,178,915	6,842,078	8,513,010	63,088,327
1911	36,362,875	4,918,520	5,460,689	6,536,164	53,278,248
1912	47,559,972	5,122,342	5,668,166	7,227,382	65,577,862
1913	49,566,720	4,793,053	5,958,173	8,921,244	69,239,190

Crushing the coal as it comes from the mine is necessary in order to produce a good and uniform quality of coke, and this is specially true as regards those coals which are low in volatile matter, as the fusing elements which are supposed to be inherent in the volatile matter are thus utilized to more advantage; besides this, if it were possible to charge coal of absolutely uniform size to the ovens, a better coke would result, as in this case all particles of coal would be completely carbonized in the same period of time. In the case of large and small lumps the former require an added carbonization period over the latter, as it requires a longer period of time for the heat to reach the center of the large lump.

There are a large number of efficient coal crushers, or breakers on the market, but in making selection of a machine of this character attention should be paid to its simplicity of design, as a multiplicity of parts usually leads to operating troubles.

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[1] The Mfg. of Coke in 1913, E. W. PARKER, U. S. Geological Survey.

Crushing the coal breaks up the slate particles, thus permitting them to be easily and intimately mixed with the coal, this even mixture leading to a regular distribution of the slate particles through the cellular structure of the coke, as, if large pieces of slate are permitted to enter into the coke, they will break the continuity of the cellular structure, this having the effect of breaking the coke into small pieces during handling. Fine crushing, say to 85 per cent. of the coal through a 1/8-in. screen, is advisable, as it is possible that the fine crushed ash constituents will tend toward a slight fusion effect in the cellular structure, this effect assisting in strengthening the coke.

Washing the coal is often, but not always, a necessity, as is seen from the foregoing tables, but as a rule such coals as do require washing also require a careful study of their physical and chemical qualities in order that the proper apparatus can be devised to secure the desired result. A proper classification of the coals ensures the most efficient separation of slate and other impurities, this being accomplished to a great extent by a difference in specific gravities of the materials charged to the washer, where coals of the same size, due to their lesser weight, will rise, while the heavier impurities (slates, pyrites, etc.) will sink. Attempts have been made to treat such coals as have their slaty impurities mixed with clay, in a dry manner, as in water washing the clay dissolves and adds to the washing troubles. This method of dry washing calls for passing the classified coals through a current of air, in which the separation is again effected by reason of the difference in specific gravity between the coal and the impurities. In order to secure the best coking results the coal should be carefully analyzed, and if upon examination it is found that costly treatment is required to secure the desired result, that particular coal should be abandoned if possible.

A great many washing systems have been and are being used, but in adopting any system care should be taken to see that it is effective for the particular coal to be treated.

At Johnstown, Pa.,[1] almost from the beginning of operations, the coal has been washed in order to reduce the ash and sulphur. The Tennessee Coal, Iron and Railroad Company also wash practically all their coals for coking purposes, the prime factor in this case being ash reduction. The coal at Johnstown is washed in bumping table washers, while in Alabama and for Illinois coals jig washers are in favor. The United States Steel Corporation proposes using a coarse and fine jig, and they have been experimenting with a wet classifier for very fine coals, experiments with the dry cleaning of coal also being under way, but only variable results with the latter have been secured as yet; MR. MEISSNER is of the opinion that further efforts should be made to study the possibilities of dry cleaning by means of air pressure or exhaust, as this system presents many advantages over the wet, if it can be made a practical and commercial success.

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[1] C. A. Meissner, Chairman, Coke Com., U. S. Steel Corporation.

It is an unquestionable fact that coal washing will greatly assist in producing a high-grade coke, as even a comparatively low ash coal will occasionally run high in ash and possibly sulphur, due to careless mining or to the irregularity of the coal seam, the washing ensuring a more uniform quality of coke, but before installing a washing system the economies of its operation must be carefully studied.

The preparation of coal in the jig, which makes use of specific gravity in the process of separation, presents many annoying and disagreeable conditions due to the physical characteristics of the material under treatment, and the many forms of the material, ranging from cubes to round, oblong, triangular, and square pieces, only add to the difficulties, as it is almost impossible to build a jig which will efficiently separate this variety of forms.

Another system of separation[1] makes use of a broad stream of material, the width being about 5 ft., so spread out that the various pieces will not touch each other in their forward movement, with an intermittent feed upon a short adjustable inclined floor, a narrow section of slate or other gritty material being located at the lower edge of the floor. A narrow opening, several inches wide, is located immediately below the section of slate, beyond which the inclined floor is continued. The operation causes the refuse in passing over the slate section of the floor to be retarded, thus dropping it through the narrow opening, while the coal, due to its less angle of repose, continues over the slate section, with only a slightly retarded velocity, and jumps over the narrow openings onto the inclined floor below. AYRES states that the serious objection to this type of machine lies in the great loss due to the chipping of the coal as it impinges against the upper edge of the continued floor below the opening, and that this loss is reported to be greater than that in the jig.

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[1] W. S. AYRES, Cong. of Applied Chemistry, Vol. 10, 1912.

A second system makes use of a spiral floor descending and inclined inwardly toward its vertical axis, depending for its operation upon the inward inclination of the floor toward the vertical axis, thus tending to draw the slate in its retarded movement toward the axis and allowing the coal, in its accelerated movement down the spiral incline, to be thrown from the floor by centrifugal force, the coal being then caught in a runway spaced a little distance away and paralleling the spiral floor. Ayres states that this device is also very destructive in its action, as every piece of the coal impinges against the outside of the runway as it leaves the spiral floor, the loss being about equal to that caused by the jig.

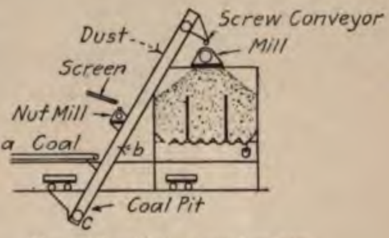


Fig. 16.—Baum coal mixer.

A third device, as described by Ayres, consists of a moving floor inclined forwardly and transversely, the motion of the floor being upward on the transverse inclination, both inclinations being adjustable, the material traveling across the floor on its longitudinal inclination. The floor, due to its upward movement, removes the impurities laterally from the forward moving stream, and the coal slides upon and across the moving floor without impact. Ayres states that the loss in this device is less than 0.5 per cent., and that it is the only machine which does not create material loss by chipping.

The mixing of high and low volatile coals in Germany is shown diagrammatically in Figs. 16 to 24,[1] where Fig. 16 represents a Baum mixer in which one grade of coal is brought from storage on the belt (*a*) and delivered to the bucket elevator (*b*), while the other grade of coal is delivered by cars to the elevator pit (*c*). If nut coal is used, the latter can be crushed in the mill and then delivered, together with any coal dust, to the elevator, from which a spiral conveyor is used to distribute the mixture to several centrifugal mills, the latter completing the mixing in a thorough manner and then depositing the coal, ready for use, in the storage bins.

[1] Dr. F. Korten, Stahl u. Eisen, 1914, page 269.

Another **Baum** plant, Fig. 17, makes use of a "dehydrating" belt for delivering the one grade of coal directly to a scraper conveyor, while the bucket elevator carries the crushed nut coal and coal dust to the same scraper, the latter then distributing the two grades to several centrifugal mills, from whence the mixed product passes to the storage bins, the coal being leveled in the latter by means of another scraper located under each mill. These two systems are not looked upon so much as real mixers, but rather as an integral part of the coal-washing system.

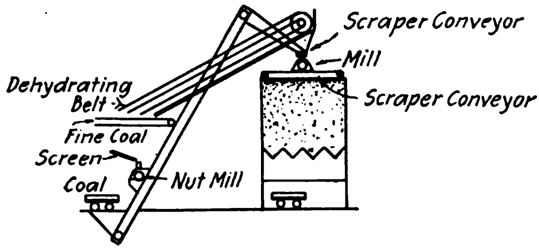


Fig. 17.—Baum coal mixer.

Fig. 18 shows a **Humboldt** system in which the two grades of coal are delivered by cars to crushers and from thence to a bucket elevator; from the elevator the coal is distributed by means of a spiral conveyor to the centrifugal mills, the coal from the latter being delivered by means of a screw to a series of revolving distributing tables, the latter placing the coal in storage.

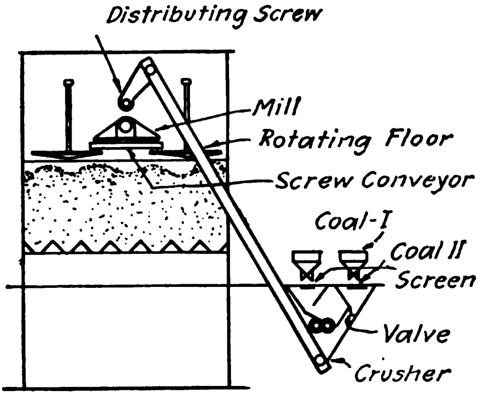


Fig. 18.—Humboldt coal mixer.

Fig. 19, which is also a HUMBOLDT system, shows the two grades of coal joining each other in the elevator boot, the elevator delivering this preliminary mixture to a screw which again distributes the coal to a series of mills from whence a scraper conveyor deposits the final mixture in the storage bins. The proper

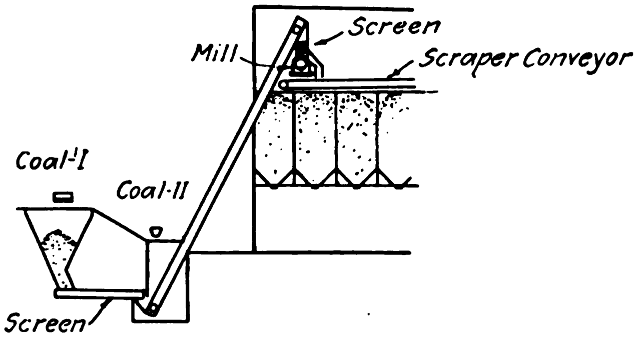


FIG. 19.—Humboldt coal mixer.

mixture is secured in these various systems by regulating the speed of the conveyors which thus permits of delivering only the amount required. The HUMBOLDT systems are also usually found in connection with coal washeries.

In the MÉGUIN system, shown in Fig. 20, the one class of coals is drawn from storage into a screw conveyor which in turn

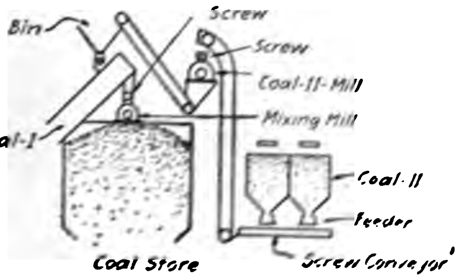


FIG. 20.—Méguin coal mixer.

deposits them in the elevator boot; the elevator places these coals in the mills, through the medium of a screw conveyor, and from the mills a second elevator carries the crushed coal to an overhead bin, it being delivered from there to the boot of a third elevator carrying the other class of coal; the two coals now pass into a mixing centrifugal mill, and from thence into storage.

The KLÖNNE system, as shown in Fig. 21, is not dependent upon coal washeries, and it is therefore found in coke plants where no washing is done. The two grades of coal are drawn from the lower storage bins, being fed to the bucket elevators by reciprocating feeders; these elevators deposit the two grades of coal in adjacent bins from whence they are drawn in measured quantities by means of reciprocating feeders to the mills, the latter delivering the mixed coal to a screw conveyor which carries it to final storage.

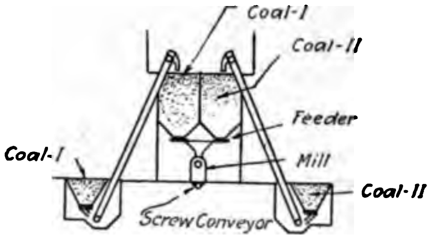


FIG. 21.—Klönne coal mixer.

Figs. 22 and 23 show the mixing systems as built by SCHÜCHTERMANN and KREMER; in the former the two grades of coal are delivered on separate tracks to the plant, dumped into pits, from whence the delivery to the mixing bin is regulated by the speed of the elevators; from this bin the mixed coals pass into the centrifugal mills, and are then deposited in the final storage by means of an elevator and scraper conveyor. In the system shown in Fig. 23 the two grades of coal are stored in adjacent bins, those for one grade being provided at the bottom with an

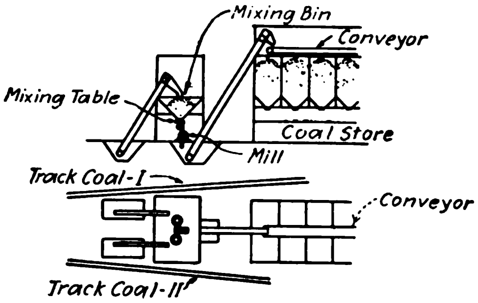


FIG. 22.—Schüchtermann and Kremer coal mixer.

ordinary gate, while the other is supplied with a reciprocating feeder, the two coals passing onto a belt and from thence to an elevator to final storage.

The CONCORDIA COLLIERY, Oberhausen, Germany, secures a desirable mixture without the use of any particular mixing system, as it was found that the various methods of delivery required by the design of the entire plant accomplished the result without any trouble, this delivery method being shown in Fig. 24. The one grade of coal is delivered by cars (*a*) to the pit of elevator (*c*), the other grade reaching the same pit through chute (*b*); the elevator deposits the mixture on belt (*d*), which in turn gives it to belt (*e*), from whence it passes into the final storage bins. These several transfers mix the coals very effectively, as tests made showed the following maximum differences between the actual and desired volatile contents at the various points of transfer:

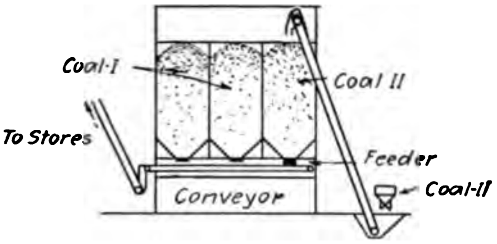


FIG. 23.—Schüchtermann and Kremer coal mixer.

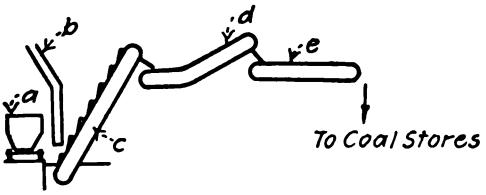


FIG. 24.—Diagram of the coal mixing plant at the Concordia colliery.

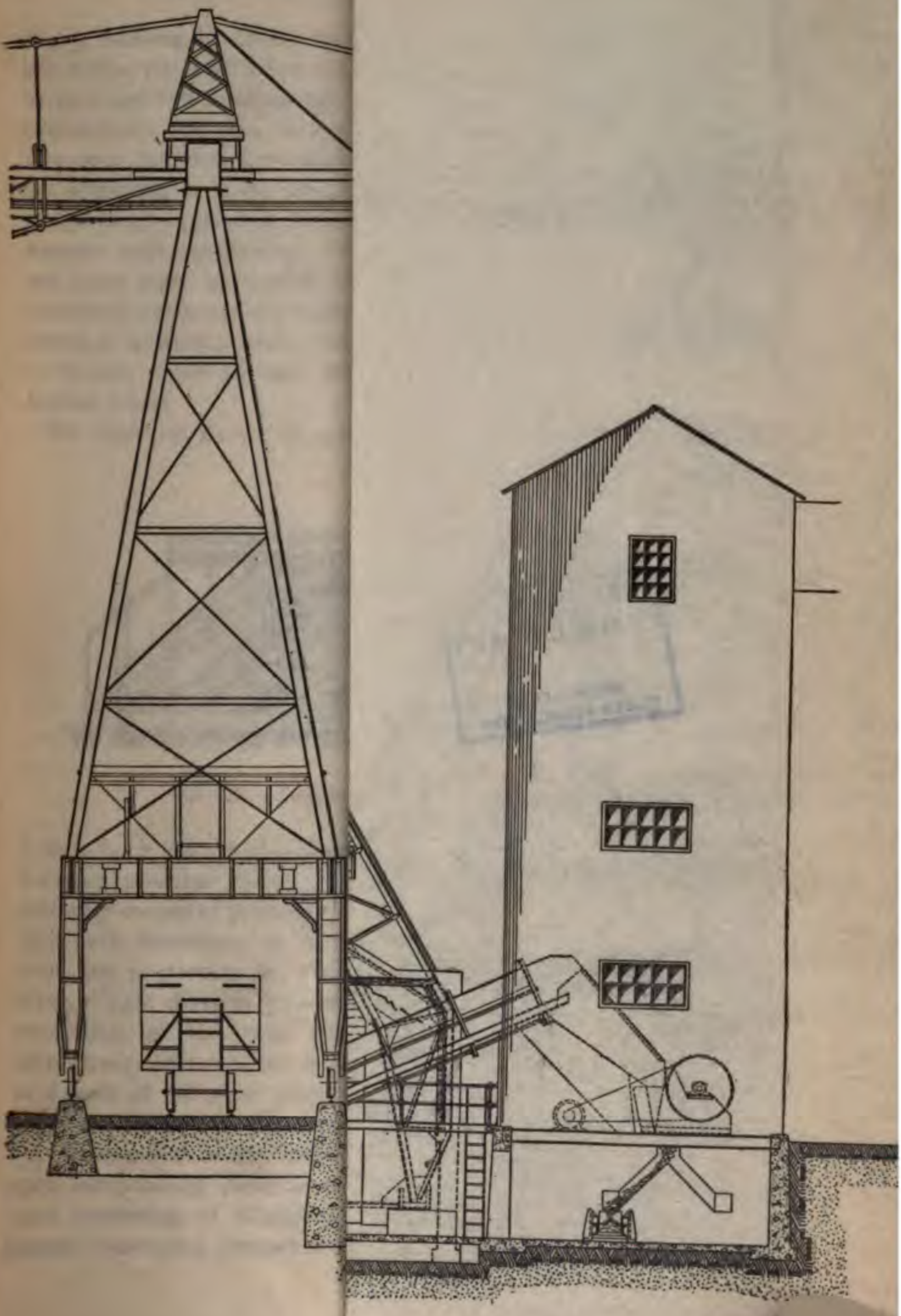
Bucket elevator (*c*)	4.87 per cent.
Scraper belt (*d*)	2.54 per cent.
Scraper belt (*e*)	1.21 per cent.
Coal storage	0.68 per cent.

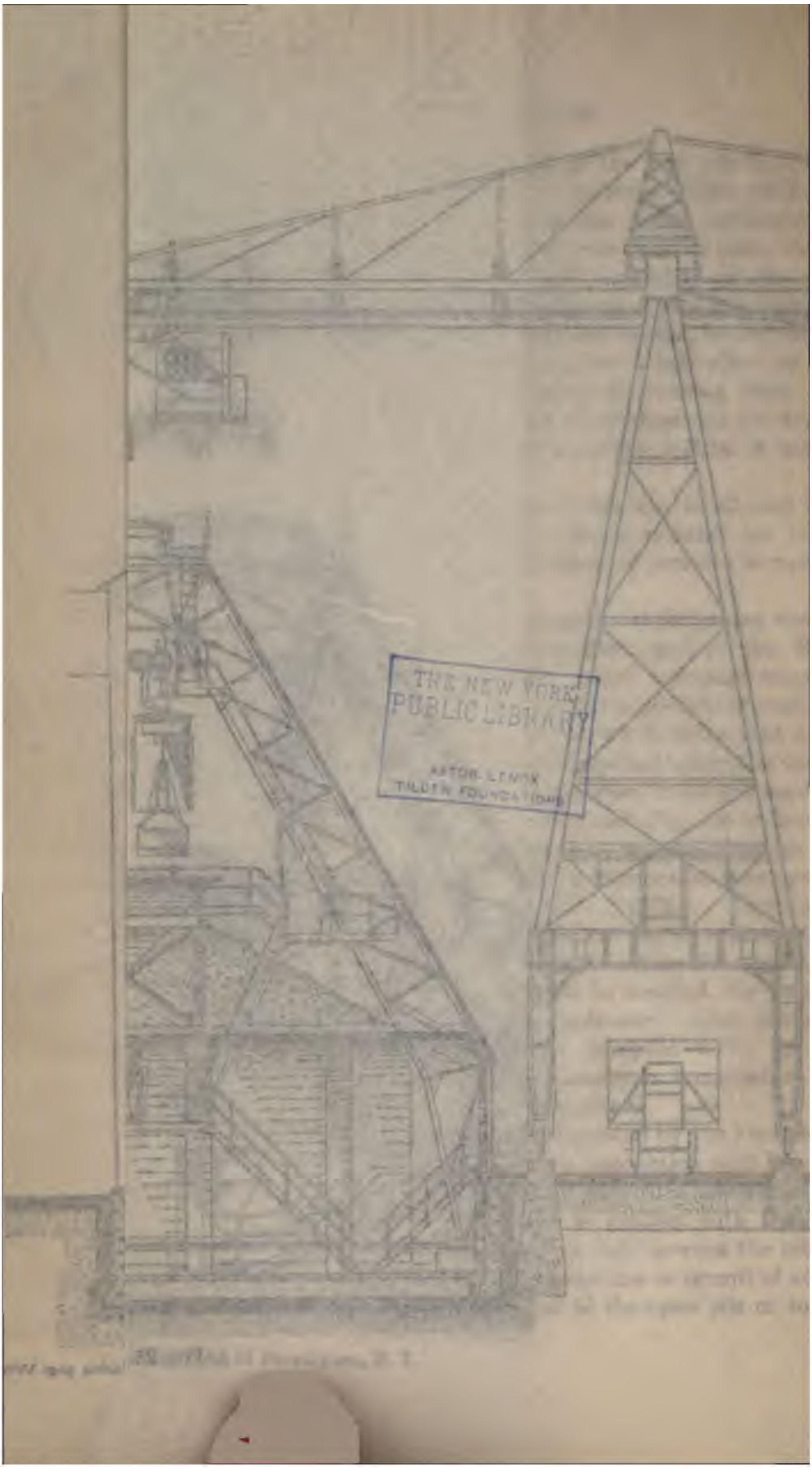
Washing the coal at the mines is very objectionable, because the drying of the coal calls for very large and costly storage bins, and even after protracted storage the coal will be found to contain a considerable amount of moisture which results in increased freight charges, this moisture being subjected to freezing during cold weather, which in turn calls for costly thawing-out installations. When the coal is washed at the carbonizing plant it is also found to contain water, and this moisture content leads to difficulties in crushing and mixing, besides which the charging of wet coal in the ovens has a bad effect on the oven walls, the water also increasing the coking time. At Joliet, Ill., a process is in the course of development for drying the coal by mechanical means, and its success will be of benefit to all coal carbonizers.

The storage of coal in open piles is familiar to all, and it is not my intention to describe the various systems for transportation and piling, but rather to confine my remarks to modern storage buildings.

A great deal of attention has been given to the proper storage of coal by the United States Government, and at the Navy coaling station on Narragansett Bay, where a storage capacity of 60,000 tons has been established, all of the pockets or coal bins have been provided with temperature tubes in order that a rise in temperature may be discovered and remedied before the danger point has been reached. These temperature tubes are made of 4-in. galvanized pipes about 20 ft. long and they are so located in the pocket floors that they project upward through the coal, each tube being provided with a thermostat arranged to indicate temperatures in excess of 65°C. (149°F.). There are a total of 232 tubes from which circuits are run to a general annunciator located in the superintendent's office; the heating of any particular pile causes an alarm to be sounded, the location of the heated pile being shown by an indicator. After extensive experiments the Navy Department accepted 65°C. as the danger point, as it was learned that increase in temperature was very rapid after this point had been reached.

The coal storage building which is operated by the Providence Gas Company at Providence, R. I., in connection with its open coal pile, consists of an A-frame steel structure, shown in Fig. 25, those portions of the structure in contact with the coal being covered with concrete, while each bay between the trusses is supplied with a steel door at the ground line to permit of access in case of fire. The coal is delivered to the open pile or to the





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storage building by means of a man-trolley operating on a movable bridge, the coal being deposited in the building through the movable hatches forming the roof. The inside of the building is provided with two man-trolleys operating grab-buckets, the latter recovering the coal from the two end sections of the building and depositing it in the large steel bin in the center, this center portion being separated from the end sections by means of two concrete walls extending entirely across the building. The coal passes from the center bin through a weighing device onto an inclined conveyor, the latter delivering the coal to the crushers located in an annex, from which the coal is elevated and carried to the bins which in turn feed the vertical retorts in the carbonizing house.

The object of a covered storage in a gas works is to prevent a loss in both the gas and ammonia constituents of the coal, and European practice dictates the separation of the coal in containers by means of partition walls which cross the storage building in both directions, so that if spontaneous combustion should occur, any particular bin may be emptied without affecting the others. This division into compartments also permits of easy supervision of the coal charged, because this arrangement affords a separation of the coal delivered at various times as well as of coals of different quality.

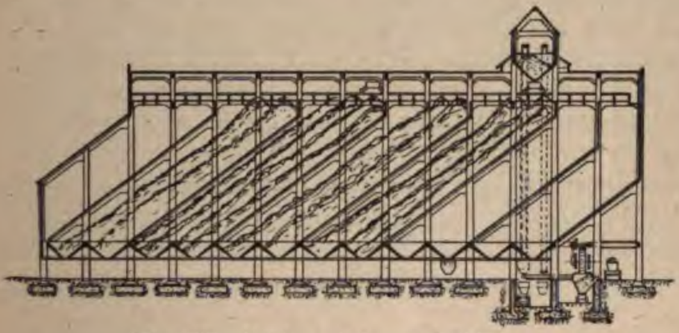

FIG. 26.—Rank coal storage building.

The coal is usually delivered to storage by mechanical means, but the selection of any method of delivery will depend entirely upon surrounding circumstances, the mechanical delivery devices consisting of telphers, elevators, belt conveyors, grasshopper conveyors, scraper conveyors, etc., the delivery devices being so arranged that the coal can either be deposited in the storage or carried direct to the point supplying the carbonizing apparatus. The storage buildings are constructed either as large halls where the coal is handled by means of grab-buckets and man-trolleys or in the shape of large silos which in turn deliver the coal through bottom chutes to the conveyors. If the coal is stored in large halls, the sides should always be so constructed as to permit of coal removal through side doors in case of fire, while if the silo construction should be adopted the delivery devices in the bottom should be so arranged as to remove the burning coal in a minimum of time.

The silo construction permits of a greater height of storage than does the storage in large halls where the coal is recovered by grab-buckets, and the silo construction as devised by **Herr Rank**, Munich, and as built by the Bamag Company, is finding great favor in Germany, this device being shown in Fig. 26. This method of storage permits of storing very large quantities of coal on a minimum of ground space, the one illustrated allowing of about 1½ tons per square foot of area, this content being due to the inclined bottoms, which construction also prevents breakage of the coal while being deposited in the various bins. The structure is built almost entirely of ferro-concrete, the space below the silos being open and fitted for the storage of other materials.

PART II. COKE

CHAPTER VI

CARBONIZATION IN RETORT BENCHES

In the process of manufacturing illuminating gas, the retort is charged with coal and then hermetically sealed by closing the door, the coal in the retort being subject to the action of a high heat, thus effectively distilling, or carbonizing the coal. The decomposition of the coal evolves gas, which escapes through the ascension pipe into the hydraulic main if a water seal is provided, or into the foul main if worked dry; if the coal has been properly distilled, the residue in the retort will be coke, but if this should not be done, a great deal of tar will be formed and the resultant coke will be of poor quality.

When the heat in the retort reaches a very dull red, or 315°C. (600°F.), the volatile matter in the coal is resolved into tar and oil but very little gas is produced, but as the heat increases the gas is evolved and the quantity of tar is diminished. Care should be taken to keep the inside of the retort free from incrustations or carbon, as a deposit of the latter prevents the transmission of the heat to the coal in great degree, thus increasing the amount of fuel required for proper carbonization.

The fuel burned in the furnace for supplying the necessary heat to the retorts is usually taken from the coke produced, the quantity of this fuel of course depending on the character of the coal carbonized, it ranging from 15 to 25 per cent. of the total coke, the amount used also depending upon the character of the furnace and the bench as well as upon the skill and attention of the stokers.

The benches used for the carbonization of coal range from the ordinary furnace bench, without regeneration or recuperation, to the highly developed regenerative and recuperative furnace benches, the retorts being either horizontal, inclined, or vertical.

A recuperative furnace is one in which the preheating is accomplished by continuous operation in one direction of passage of the substance to be heated (air or gas) through a flue or set of flues, and without reversal of direction, the transfer of heat taking place through the walls of the flue or flues, while a regenerative furnace is one in which a chamber or set of chambers is used for preheating the gas or air, the preheating being distinctly an intermittent operation and being accomplished by heat previously stored in the chamber or chambers.

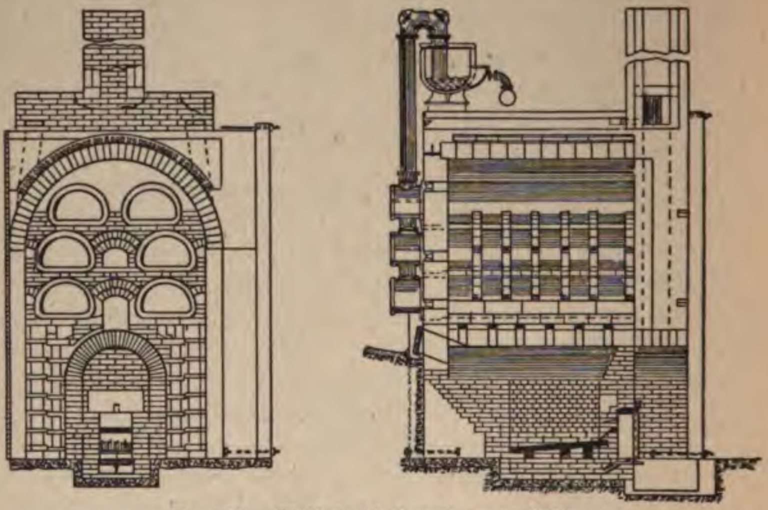

FIG. 27.—Half-depth bench, rear clinker.

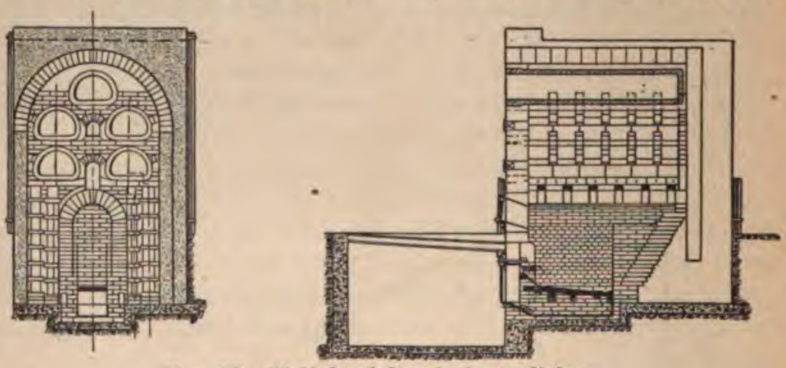

FIG. 28.—Half-depth bench, front clinker.

Fig. 27 shows a "half-depth" bench arranged to clinker from the rear, while Fig. 28 shows a similar bench to be clinkered from the front. Fig. 29 shows a three-quarter depth bench arranged for rear clinkering, and Fig. 30 shows a similar bench clinkered

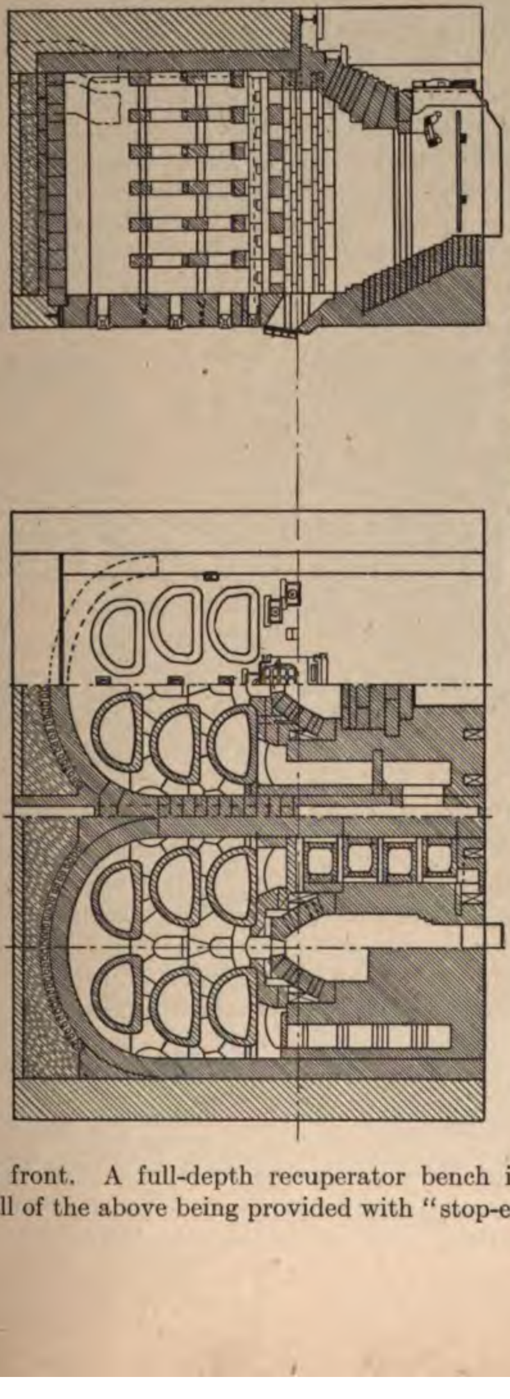

FIG. 29.—Three-quarter depth bench, rear clinker.

from the front. A full-depth recuperator bench is shown in Fig. 31, all of the above being provided with "stop-end" retorts, while Fig. 32 is a section through a bench provided with "through" retorts.

Modern systems of carbonization are now classified into "layer" and "mass" carbonizing, the former possessing a free space above the body of the coal, thus permitting the gas as it is evolved to be cracked up through the radiant heat from the crown of the retort, this cracking up of the gas assisting in the production of free carbon, naphthalene, and sulphur impurities. MR. E. G. STEWART[1] states that the temperature of the gas leaving the retort in the layer system ranges between 315°C. and 415°C. (600°F. and 800°F.), while in the average mass system this same temperature is from 149°C. to 205°C. (300°F. to 400°F.), and that in all layer systems we find a much lower percentage of tar oils condensing at a comparatively low temperature than in the mass system, while free carbon and hydrocarbons with higher carbon percentage are found in appreciable proportions. The influence of the temperature of the retort is readily shown in a proportionate rise or fall in the temperature of the issuing gas and in the amount of hydrocarbon degeneration in the free space above the coal, and while a high temperature is more or less imperative, the disadvantages of this high temperature may be combated by regulating the weight and duration of a charge; this temperature is also often increased by leaks in the retorts operating under a vacuum, as gases having a tem-

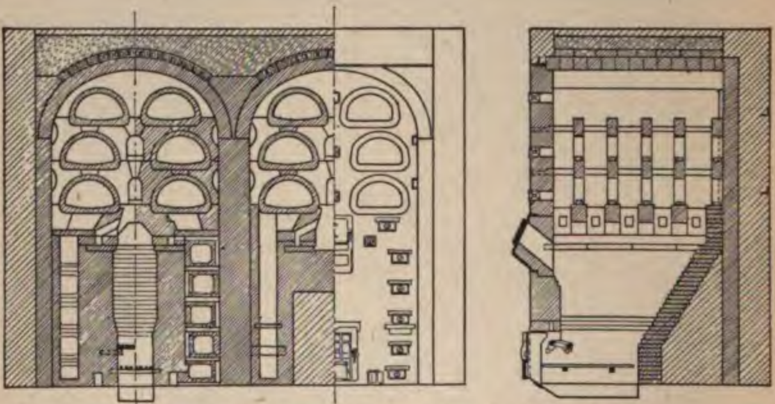

FIG. 30.—Three-quarter depth bench, front clinker.

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[1] London Jnl. of Gas Ltg., 1913, page 329.

perature as high as 1095°C. (2000°F.) are thus drawn in, this addition also increasing the gas volume.

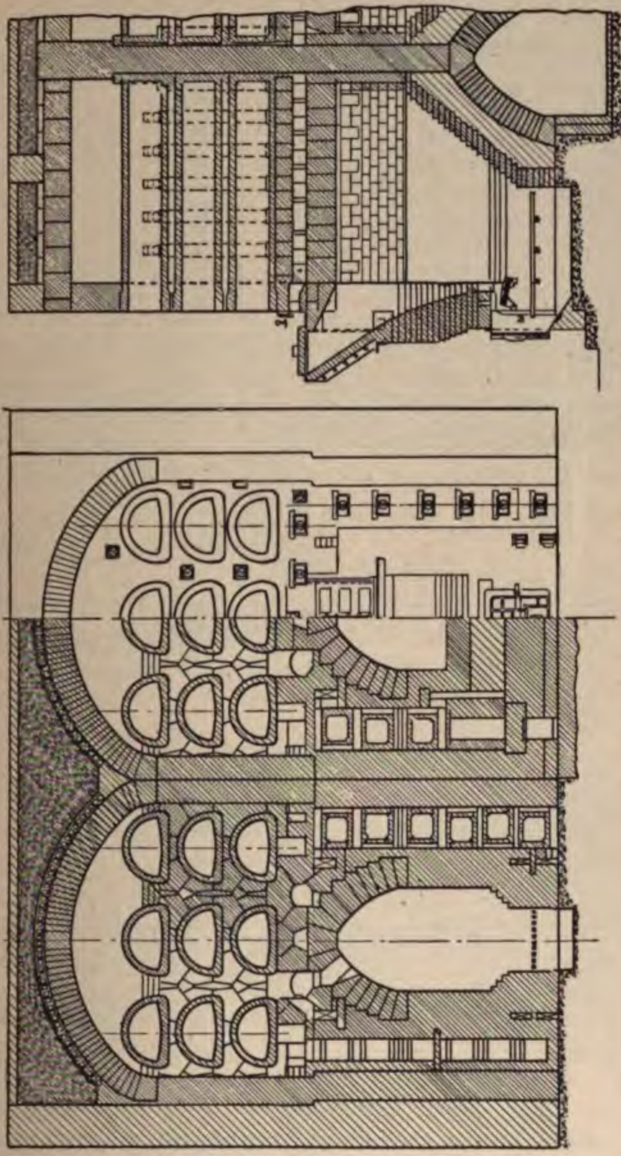

FIG. 31.—Full depth bench, stop end retorts.

Under normal conditions, the gas usually reaches the bridge pipe at a temperature of about 93°C. (200°F.), the minimum at the beginning and end of the charge being about 38°C. (100°F.), the maximum 2 hours after charging being about 132°C. (270°F.), these changes in temperature leading to the condensation of some of the tar vapors, the resultant oils running down the interior of the ascension pipe and being redistilled when they

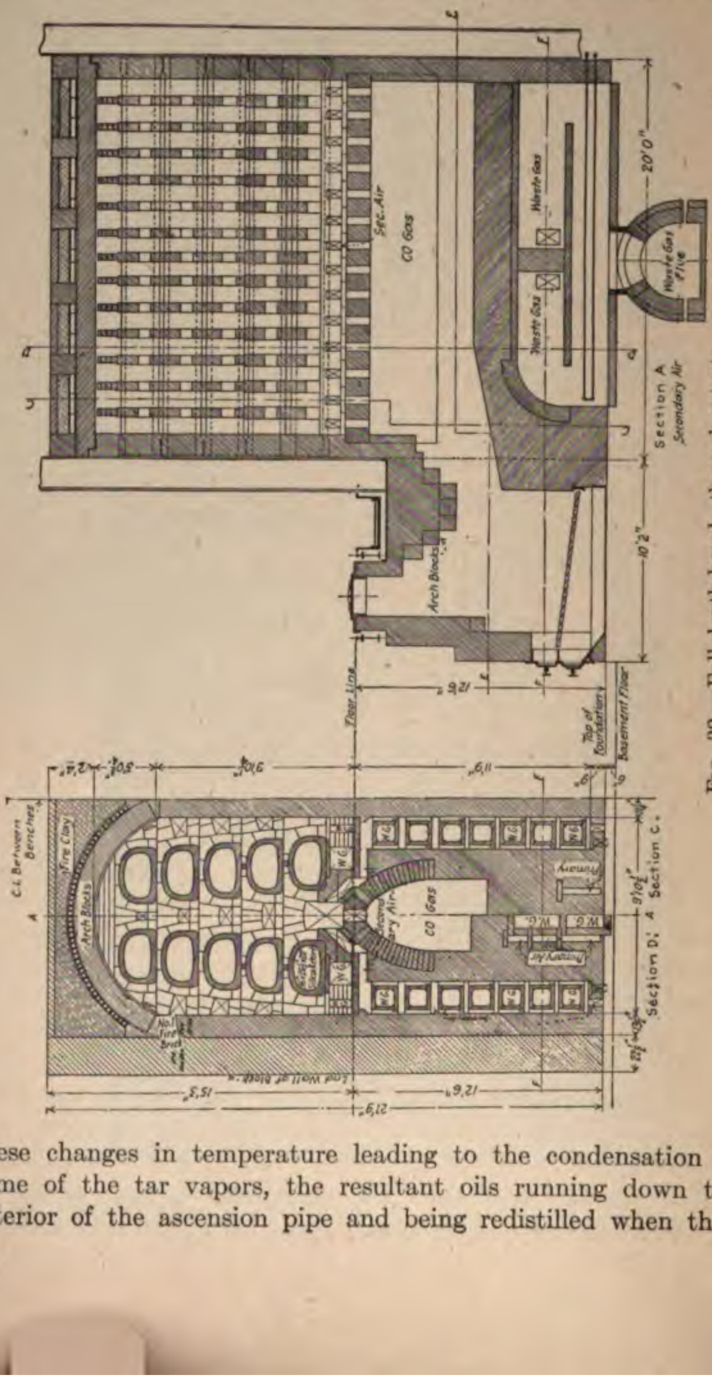

FIG. 32.—Full-depth bench, through retorts.

reach a higher heat zone. If the production of gas is uniform this condensation and redistillation will also be uniform, and under proper conditions the surplus tar thus formed will run into the mouthpiece, thus clearing the ascension pipe when the gas production decreases, this latter condition being exemplified with heavy charges and high temperatures, or with light charges and low temperatures.

In theory these small charges with high temperatures cause a decomposition of the heavy hydrocarbons and tar vapors as explained above, but practice has shown that the separation of the carbon is so energetic that the ascension pipes usually stop up, necessitating the drawing of the charge before the coal has been entirely carbonized, thus producing not only a loss in gas but also in coke. Large charges, with proper stoking machinery, can be laid more evenly than light ones, and as a longer period is required for the carbonization of the coal, the retorts will not be opened as frequently as with light charges, this leading to a longer retort life; furthermore, as decomposition is avoided to a great extent by the use of heavy charges, higher working temperatures become possible.

MR. G. M. GILL[1] states that opinions differ as to the effect of using heavier charges, but that in his experience he has found that the more fully the retort is filled, the better will be the utilization of the radiant heat, his experience being confirmed by S. CARPENTER, who found that he was able to carbonize 15 per cent. more coal in a given time when the retorts were fully charged than when they were half full; also that in regard to gas yield he has found that with heavy charges requiring from 10 to 12 hours for carbonization the make per ton of coal was considerably increased, this increase possibly approximating 500 cu. ft. per ton. GILL also states that his experience leads him to believe that the candle power of the resultant gas is lower with heavy charges, as when working with 6- and 8-hour charges the average illuminating power was 16.6 candles, while with heavy charges he secured only 15.5 candles, a drop of 6.6 per cent. GILL doubts whether the increased quantity of gas obtained by using heavy charges is of value, as its chief constituent is hydrogen, with both a low candle and calorific value, explaining this condition by the statement that when the core of the charge is thoroughly carbonized, the outer surface must necessarily be over-carbonized,

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[1] London Jnl. of Gas Ltg., 1909, page 738.

the beginning and end of the charge b
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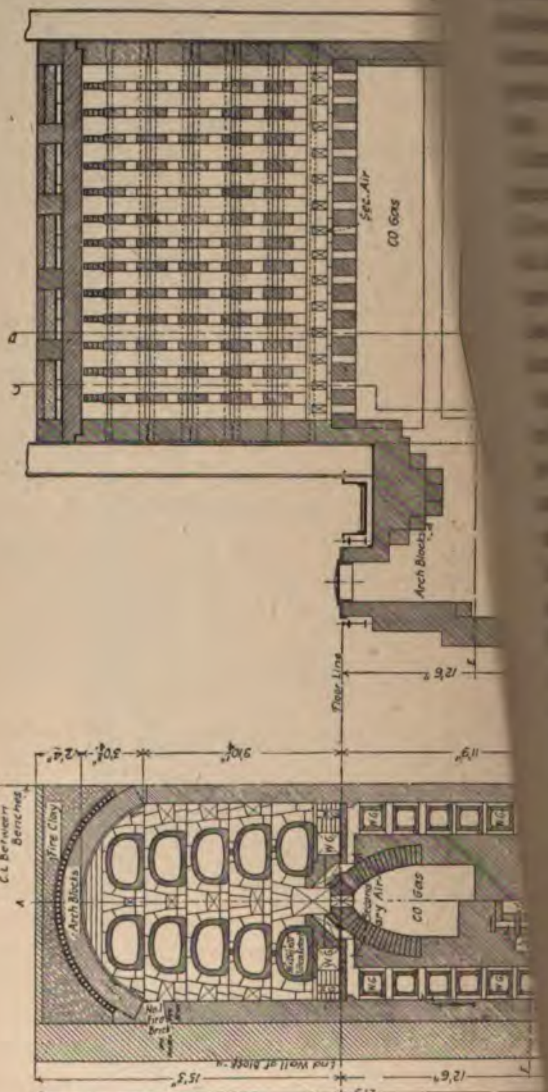

these changes in temperature leading to the
some of the tar vapors, the resultant oils run
interior of the ascension pipe and being redistil

Baehr states that this tabulation shows only an arbitrary relation, and that the extent to which recuperation should be carried in order to secure the greatest commercial economy is as yet unknown.

Mr. A. H. Strecker gives the following additional figures in this connection:

Type of furnace	Heating surface of recuperator in square inches	Ratio of recuperator surface to retort surface, per cent.
Klönne	30,720	33.9
Parker-Russel	30,240	33.3
Newark	37,152	41.1
Average	32,704	36.1

The waste gases greatly exceed the excess air in quantity, and also because of the thermal capacity it would seem almost impossible to utilize all of the available heat in the gases as they leave the setting; added to this, we find certain physical conditions which make an attempt at heating the primary air to any extent out of the question. **Mr. Baehr** states that in his opinion the ideal setting would include separate but continuous gas producers, so arranged as to supply a stack of benches with combustible gas, making as much use as possible of the sensible heat of the waste gases for preheating the secondary air, and the balance for heating the primary air.

If the bench is provided with its own producer, the latter should have a rectangular cross-section, but if separate producers are supplied, they should be of a circular shape.

Mr. R. W. Polk, in defining the character of a setting, states that a half-depth bench is one in which the distance from the grate bars to the crown of the furnace is 36 in.; for a three-quarter depth this dimension is 54 in.; while for a full depth it should be 72 in., it being understood that in all cases the recuperators extend to the level of the top of the ash pan. As regards the heating of the primary air, he states that this is uneconomical, as this procedure merely means the carrying of more heat into the furnace where it should be the aim to keep the temperature down. **Baehr** does not agree with the above definitions, and offers the following: A shallow producer recuperative bench is one having an average effective fire depth just before firing of not less than 3 ft., and a maximum of not over 5 ft.; the ratio of effective recuperative surface to effective interior surface of all retorts being not less than 40 per cent., and the ratio of effective cubical contents of all retorts in cubic inches to the effective recuperative surface being not less than 9 per cent. A medium depth producer recuperative bench is one having an average fire depth just before firing of not less than 5 ft. or a maximum average effective fire depth of not over 7 ft., the ratio of effective recuperator surface to effective interior surface of all retorts being not less than 50 per cent., and to effective cubical contents of all retorts, in square inches and cubic inches respectively, not less than 10 per cent. A deep producer recuperative bench is one having an average fire depth just before firing of not less than 7 ft. and a maximum effective fire depth of not over 10 ft., the ratio of effective recuperative surface to effective interior surface of all retorts being not less than 60 per cent., and to effective cubical contents of all retorts in square inches and cubical inches respectively, not less than 11 per cent. The author deems MR. BAEHR's conclusions correct, but as these figures are perhaps arbitrary, they are subject to further study.

In the operation of the bench, cleaning the producer should be avoided as much as possible, but care should be taken not to allow the excessive formation of clinker, as the latter is a most disturbing element in bench firing; clinkering not only injures the brick lining of the furnace, but it also reduces the heat, in consequence of which quite some time is consumed in bringing the furnace back to a normal condition.

Horizontals at Cambridge, Mass.—The installation at Cambridge, Fig. 33, consists of two batteries each of ten benches with eight 16-in. × 20-in. retorts 20 ft. long. As originally built, each bench was provided with two furnaces, this arrangement being later changed to one furnace. DE BROUWER machines are used for charging the retorts, and the plant was at first operated on 6-hour charges with high heats; this mode of operation produced some sluggish tar during cold weather, and since then the plant has been operated at a lower heat, but with the same coal charges, the carbonizing period being increased to 8 hours, this change in operation producing better working conditions and a larger gas yield. The average results at this plant are given below:

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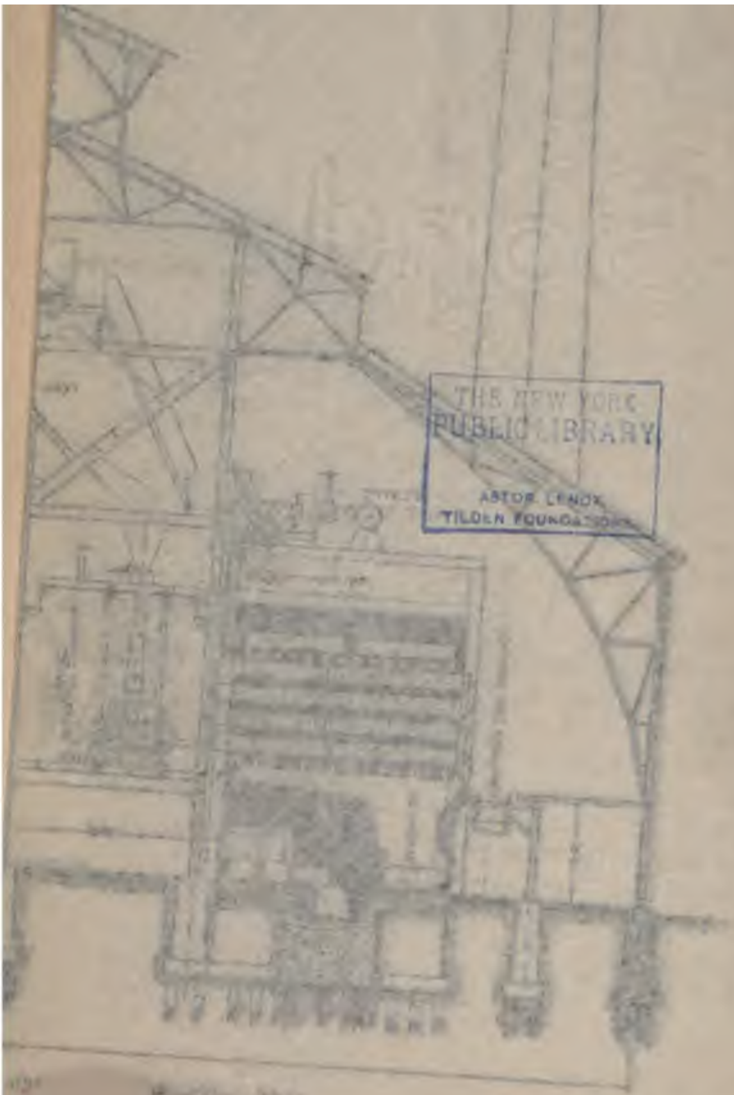

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Year	Yield per lb. cu. ft.	C. P. Sugg.	Candle ft.	Average cu. ft. per retort	Generator fuel, per cent.
1909	5.16	15.44	79.71	24,250	13.09
1910	5.55	15.25	84.64	21,980	13.12
1911	5.57	15.55	86.70	20,310	10.96
1912	5.55	15.30	87.90	20,540	11.20

Horizontals at Stuttgart, Germany.—This installation consists of 22 benches each containing nine retorts 15.7 in. × 23.6 in. × 19.7 ft. long, De Brouwer machines being used for charging and discharging. The charges consist of 6½ tons, this amount being burned off in 9 hours. The average make is 11,698 cu. ft. per ton (1909), the gas having a net calorific power of 560 B.t.u.'s, while the average make per bench per day, including scurfing time, was 155,390 cu. ft., corresponding to 164 cu. ft. per square foot of retort house ground area per day. The coke produced is large and firm, the coke used in the furnace being from 13 to 14 per cent. by weight of the coal carbonized.

Horizontals in England.—Dr. J. Ferguson Bell[1] reports simultaneous tests extending over a week on carbonizing with full horizontal retorts and with light charges, the respective carbonizing periods being 12 and 8 hours, the retorts measuring 16 in. × 22 in. × 18 ft. long; the coal used was Derbyshire Black Shale of ordinary quality.

Composition of Coal Used

Carbon	72.22
Hydrogen	4.75
Nitrogen	1.35
Sulphur (vol.)	2.31
Ash	7.51
Moisture	4.49
Oxygen (by diff.)	7.36

Analysis of Coke Produced

Charge	Sp. gr.	C	Vol.	Total S	Ash	Moisture
12 hours	1.59	84.82	1.838	2.780	10.34	0.664
8 hours	1.63	84.05	1.021	2.415	10.11	0.290

The coke from the 12-hour charge was somewhat harder and of better color than the other.

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[1] London Jnl. of Gas Ltg., 1910, page 836.

A summary of the test results is given below:

	12 hours	8 hours
Average weight per charge, lb	1,344.0	920.0
Gas made per 2240 lb., cu. ft	12,091.0	11,820.0
C. P. Metropolitan No. 2 burner	15.56	16.29
Calorific value, gross, B.t.u	608.2	586.8
Calorific value, net, B.t.u	554.6	530.4
CO_2 in crude gas, per cent	2.5	2.4
H_2S in crude gas, gr. per 100 cu. ft	1,040.0	933.0
CS_2 in crude gas, gr. per 100 cu. ft	65.9	57.3
Coke sold per ton of coal carb. large, lb	1,053.0	997.0
Coke sold per ton of coal carb. small, lb	277.0	331.0
Coke sold per ton of coal carb. breeze, lb	87.0	83.0
Coke used in furnaces per ton of coal carb., lb	179.0	185.0
Coke used in furnaces per cent. of coal carb	8.0	8.25
Tar made per ton of coal carb. gal	12.07	10.43
Specific gravity of tar	1.156	1.189
NH_3 liquor per ton of coal, 50°Tw. gal	33.53	33.88
NH_2 in lb. of ammonium sulphate, lb	29.89	28.50
Analyses of Gas:		
Carbon dioxide	2.20	2.60
Illuminants	3.80	3.30
Oxygen	0.30	0.30
Carbon monoxide	5.60	5.20
Methane	32.90	34.00
Hydrogen	48.60	43.50
Nitrogen (by diff.)	6.60	11.10
Calorific value, calculated, B.t.u	567.6	550.2
Calorific value, by JUNKER's calorimeter	566.0	515.2
Temperature in retorts, mean average, °C	995.0	986.0
Temperature in retorts, mean average, °F	1,823.0	1,807.0
Temperature in combustion chamber, °C	1,332.0	....
Temperature in combustion chamber, °F	2,430.0	....

The Carpenter Vertical Take-off.—This take-off, U. S. patent No. 1,122,683, has been in operation for a little over 2 years, and has proven the elimination of a great deal of the labor found in connection with the usual ascension pipe. As originally constructed, this take-off was provided with a circulating system, the ascending gas being cooled by the introduction of water at the top of the pipe. After operating for some time, however, it was found desirable to maintain as high a temperature as would be practicable in the pipe, in order to secure a fixing action for the large volume of low-temperature gas coming from the freshly charged retort, and consequently in new constructions no water will be introduced into the pipe. The theory has been advanced that if a sufficient quantity of circulating liquid were passed through a pipe of this nature, tendencies toward stoppages or formations on the sides of the pipe would at all times be eliminated, but careful consideration and observation has proven to the manufacturers that if an economical carbonizing temperature is maintained within the retorts, this pipe must be given a certain amount of attention to prevent any inner formation, even though large quantities of liquid are circulated. The advantage to be gained by a thorough mixing of the gases issuing from the various retorts, and their travel through this pipe at a comparatively high temperature, is thought greater than that resulting from any tendency toward elimination of formation inside the pipe by the process of cooling the gas by means of a circulating medium. In order to reduce the necessary attention to these pipes to a systematic operation,

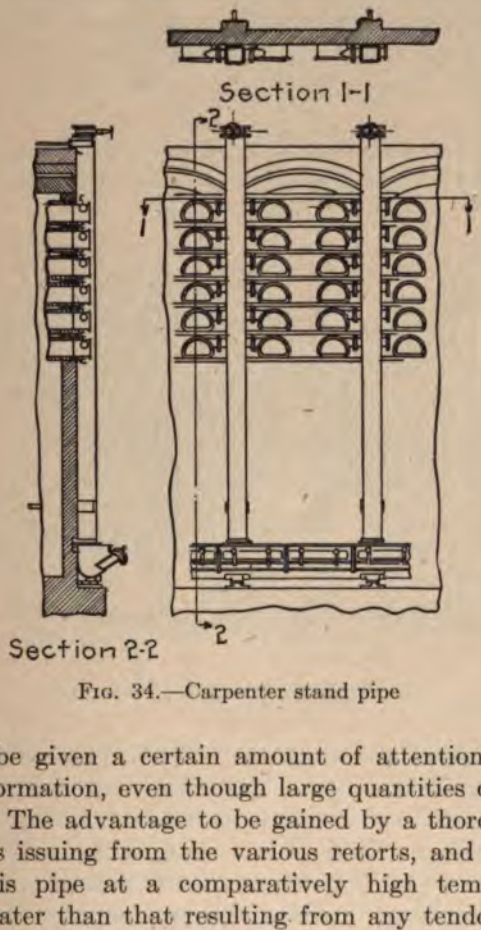


FIG. 34.—Carpenter stand pipe

the manufacturers are now providing a cleaning apparatus especially designed for this purpose. As will be seen from Fig. 34, the top of each vertical pipe is closed with a gate valve, while the lower end is connected to a tar main. A cleaning compartment, containing the cleaning tool, mounted on an electric hoist directly over the line of pipes, is so arranged that it can be connected to the top flange of the gate valve by means of self-sealing clamps, after which the gate valve is opened and the cleaning tool lowered through the pipe, any formation formed being ejected from the bottom of the pipe. This method of cleaning does not require the opening of the pipe to the atmosphere, and the attendant is not exposed to escaping gases or undue heat.

The Congdon Scrubber Standpipe.—In this system, Fig. 35, U. S. patent No. 1,099,639, the opposite principle to that adopted for the Carpenter pipe is used, in that water is injected into the top of the pipe. The standpipes are rectangular in cross-section and are arranged on either side of the buckstays, the two standpipes from each bench sealing above in a common wash box, while the lower ends dip into a tar trough. A pump located on the lower floor forces liquor from the tar trough into the overhead supply tank, from whence the liquor flows through various funnels into and through the standpipes back to the lower seal. Arrangements are also made to supply city water or

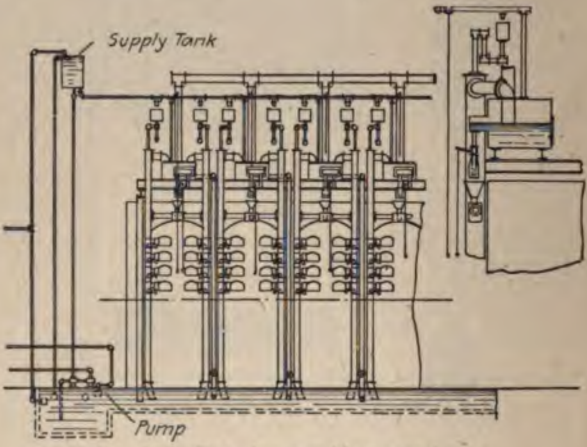


FIG. 35.—Congdon scrubber stand-pipe.

water from the condensers to the supply tank in case liquor from the lower seal is not to be used.

Fig. 36 shows one type of inclined retort bench, the intermediate step between horizontals and verticals; very few benches with inclined retorts are being built in the United States, the general practice approving of horizontals, but opinion in a great many instances is now veering toward the verticals, of which type the BUEB, or DESSAU, the GLOVER and WEST, and the WOODALL-DUCKHAM are the principal exponents. Fig. 37 is a cross-section through the Astoria inclined retort house.

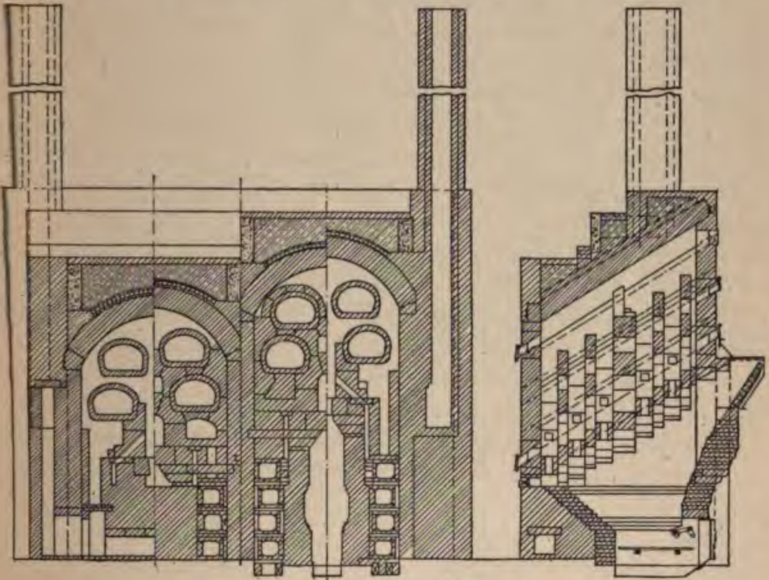


FIG. 36.—Inclined retort bench.

The introduction of the vertical retort was attended with a great many difficulties, and this in spite of the seeming simplicity of its operation, as it would appear that the introduction of coal into the top of the retort, and the discharge of the coke from its bottom is the most rational method of carbonization, but many of the original failures with this type of retort can most probably be traced to lack of knowledge of the difficulties involved.

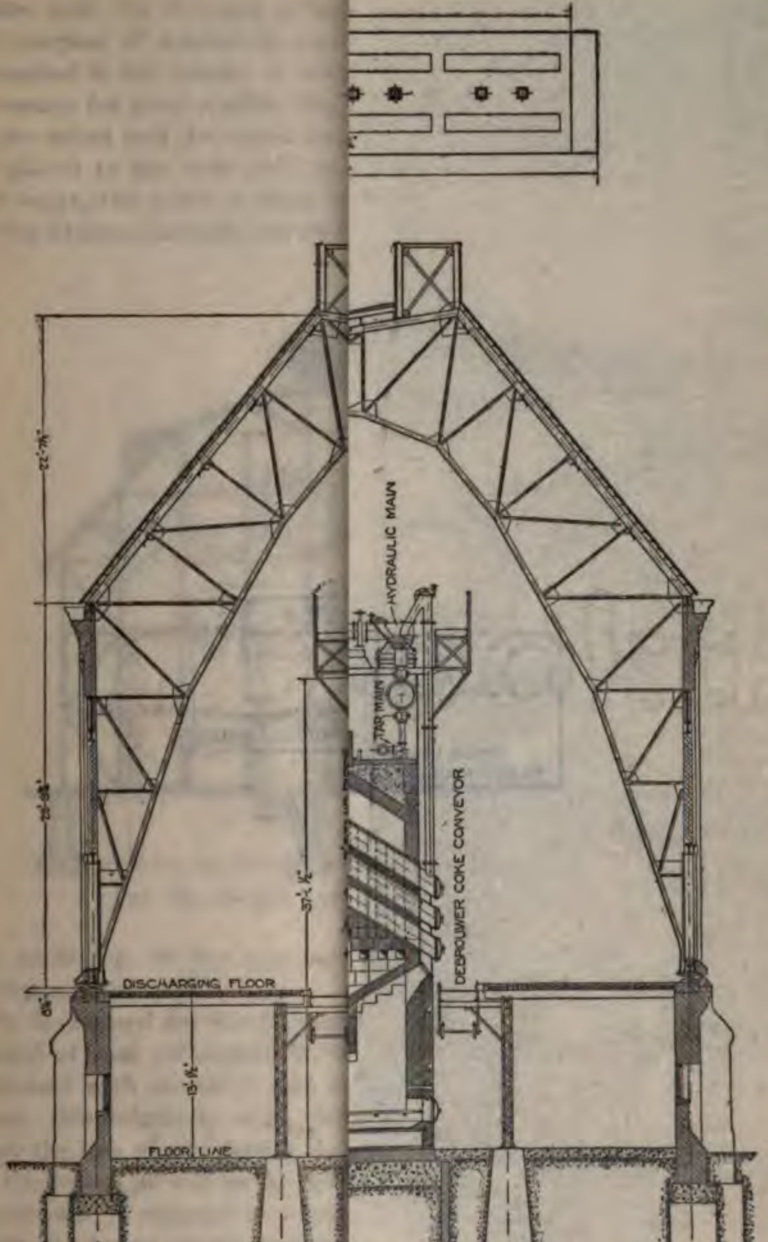
The Dessau Vertical Retort.—The first BUEB verticals were installed at Dessau, Germany, hence their name, and after some degree of commercial success had been obtained at this plant they were introduced at others, with the result that in many instances their installation practically revolutionized bench operation.

The Bueb, or Dessau, bench is built to contain eight, ten, or more retorts, varying from 13 to 16 ft. in length, generally arranged in two rows, side by side. The generator which supplies the necessary heat to the retorts is located on the longitudinal side of the bench, while the recuperators are placed on each side of the generator or producer. The top of the retort is closed with an ordinary mouthpiece and self-sealing lid, while the bottom discharging doors are so connected to a mechanical device as to permit of operating all of the doors of any one bench from one central point. The retorts are heated by ascending gases, these gases being conducted from the top of the bench in a downward direction through the recuperators, and from thence to the main smoke flue. Coal is charged to the retorts through the medium of suspended coal hoppers running on tracks located above the benches, the coke being discharged by opening the bottom doors and thus permitting the coke to slide over a movable chute into a conveyor located convenient to the bench.

The later designs of Bueb benches contain three rows of retorts, the lower mouthpiece lids of the three being mounted on one shaft, thus permitting the discharge of the three retorts at one time, they also being charged simultaneously by a new arrangement of charging hopper, this arrangement practically making the three retorts one unit. These discharge mouthpieces are operated by hydraulic power from a central station, the opening and closing of the lids being accomplished by simply turning a cock. The upper mouthpieces are connected in multiple by means of a common escape pipe into the hydraulic main. It is claimed that this arrangement requires that the bridge-pipes and pitch pans be cleaned only once in from 8 to 14 days.

In charging the retorts a fixed amount of coke breeze is first deposited in the bottom in order to cover the space not affected by heat, after which the coal is placed, the coal hopper being so arranged that the quantity of coal charged can be adjusted, thus permitting each retort to be filled to the same height; a separate suspended hopper is provided for charging coke into the furnace.

The arrangement as described above is the one usually applied, because in most cases the fuel charged is a mixture of fine and



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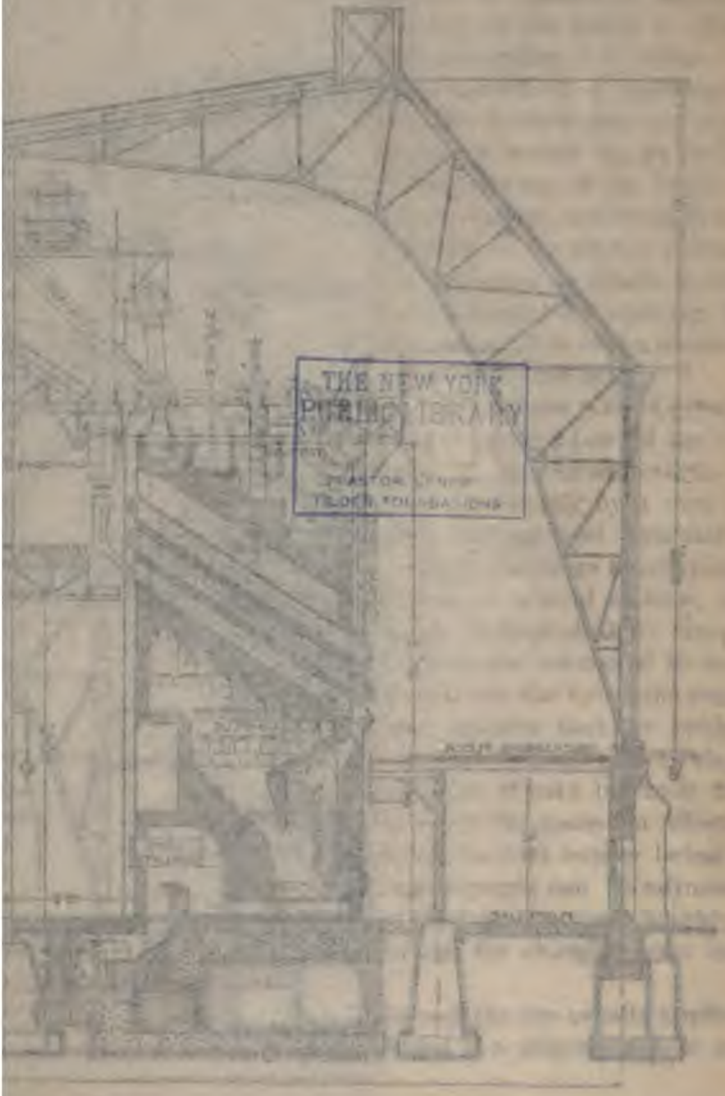

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coarse coal; the charging hopper is supplied with a partition for the purpose of separating coarse coal from the fine until it is deposited in the retorts, it being claimed that this separation is necessary for good results; the fine coal is placed at the bottom of the retort and the coarse coal at the top, or the fine coal may be placed to one side and the coarse coal to the other side of the retort, the latter method being preferred by the builders.

The Dessau installation at Providence, R. I., is shown in Fig. 38, while Fig. 39 is a cross-section through the retort house at Cologne, Germany.

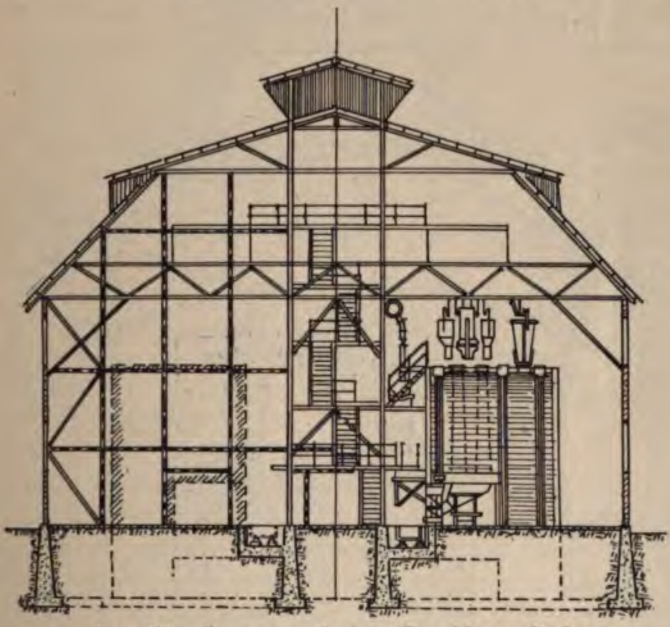

Fig. 38.—Dessau verticals at Providence, R. I.

It is claimed for the Dessau bench that the yield of gas per pound of coal carbonized is considerably higher than the yield obtained with the same coal in either horizontal or inclined retorts, this relatively high yield of gas being due to the fact that the loss of gas during the charging and discharging period of the retort, as well as the decomposition of the gas under these conditions, is reduced to a minimum; also the water vapors developed during carbonization participate almost entirely in gas production, while the quality of the illuminating gas produced in vertical retorts is equivalent to that produced horizontal or inclined retorts, and the heating value of the g

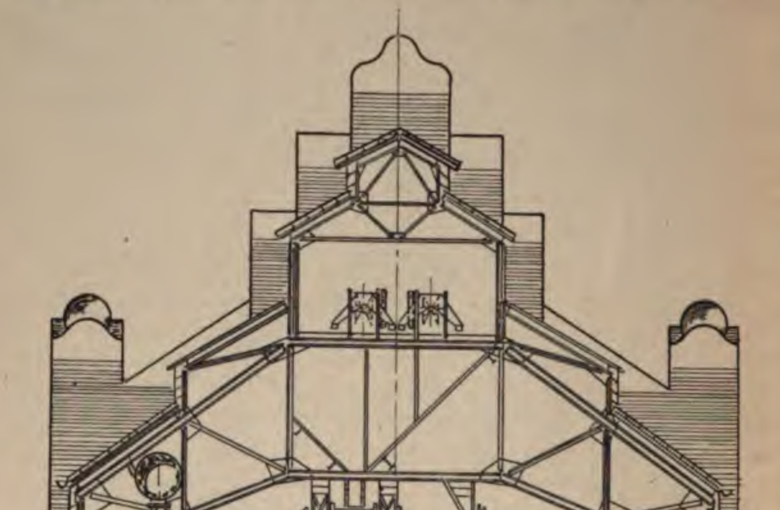

FIG. 39.—Dessau verticals at Cologne, Germany.

ranges from 585 to 630 B.t.u.'s, depending upon the character of the coal carbonized.

It is also claimed that the gas produced in these verticals contains but traces of naphthalene, and that naphthalene stoppages are entirely eliminated.

Reports from Mariendorf and Dessau, Germany, are to the effect that the amount of carbon bisulphide in the gas produced in these verticals is about 50 per cent. less than with gas produced in horizontal or inclined retorts; this reduction in carbon bisulphide is, from a hygienic standpoint, very important, because it is transformed into sulphurous acid at the point of gas combustion, and once formed, it is a difficult matter to remove it from the gas.

It is also claimed that the Dessau verticals are responsible for a higher ammonia yield than is possible with other benches, this increase amounting to from 30 to 50 per cent. The comparative yields of ammonia in horizontal, inclined, and vertical retorts, using the same coal, at the Berlin Gas Works, are given in Table XXXI.

Table XXXI.—Ammonia Yields

Yearly average for 1908	Lb. of NH_3 per net ton of coal
Horizontal benches with charging and discharging machines..	4.8
Inclined benches	5.4
Inclined and vertical benches	5.8
Vertical benches only	6.2

No claim is made for increased tar yield, the yield being from 4 to 5.6 per cent. by weight of the coal carbonized, but the quality of the tar is said to be superior, the tar being in the shape of a brown oil which does not thicken in the hydraulic or tar mains. A distillation test made on the tars produced from the same coals in vertical and in horizontal retorts, gave the following analyses:

Table XXXII.—Tar Analyses

Constituent	Vertical retort, per cent.	Horizontal retort, per cent.
Ammonia liquor	2.17	3.50
Light oil	5.85	3.10
Medium oil	12.32	7.68
Heavy oil	11.95	10.15
Anthracene oil	15.96	11.54
Pitch	49.75	62.00
Residue	2.00	2.03

It is claimed that the vertical retort tar contained from to 4 per cent. of free carbon, as compared with from 20 to per cent. in horizontal or inclined retort tar, and that the naphthalene content in the former was about 50 per cent. less tha in the latter.

The cyanogen produced in these retorts is said to be abou 40 per cent. less than in horizontals, but this commercial los will be insignificant if the ammonia yield is increased as claimed.

In the matter of gas, it is claimed that the vertical retort production is materially higher for the same length and sectional area of retort than is possible with either the horizontal or inclined bench; this condition may be explained by the fact that a better utilization of the heating surface is effected in a vertical bench, due to the complete filling of the retorts. The daily production of gas in a 13-ft. retort is from 13,500 to 14,500 cu. ft., dependent upon the character of the coal carbonized. It is claimed that these figures represent normal operating conditions, and that in case of necessity a much higher yield of gas can be obtained. At the Zurich gas works, with 100 retorts each 13 ft. long, and with a carbonizing period of 8½ hours, 1,700,000 cu. ft. of gas were produced, this being equivalent to a retort production of 17,000 cu. ft. in 24 hours.

The producing capacity of a DESSAU vertical bench depends, of course, upon the number of retorts in the bench, their length, the carbonizing period, and the character of the coal, but this yield can readily be increased by about 10 per cent. if steam is introduced into the lower end of the retort toward the end of the carbonizing period, thus adding water gas to the gas already produced; the steam is thoroughly decomposed during its upward travel through the deep bed of incandescent coke, thus also converting any remaining tarry constituents into illuminating gas. It is claimed that the heat energy consumed in the formation of the water gas is taken principally from the incandescent coke, while the carbon required is mainly supplied from the graphite deposit on the retort walls, it being claimed that this procedure also reduces the quantity of graphite which would otherwise have to be removed from the interior surface of the retort walls by some mechanical means.

This production of water gas causes a slight loss in calorific value, but it is claimed that it has a favorable effect upon the fuel consumption, because toward the end of the carbonizing period, when the production of coal gas is greatly decreased, the heat, which otherwise would be wasted, is effectively applied in the production of water gas. The fuel consumption for the production of 1000 cu. ft. of straight coal gas in the DESSAU vertical is from 27 to 30 lb., this amount being reduced to from 28 to 26 lb. with the production of water gas.

All authorities, however, do not agree as to the economy of producing water gas in verticals, as from the figures prepared by HERR PRENGER,[1] after taking into consideration all circumstances and conditions, and making all possible allowances, it would seem that the cost of water gas thus made is 64 per cent. greater than when made in special generators.

DR. R. GEIPERT[2] arrives at a somewhat different conclusion, and he gives the following comparative estimate on the cost of producing 141 million cu. ft. of water gas by means of an independent plant, and by admitting steam into the retorts.

INDEPENDENT PLANT

6 per cent. interest and depreciation on £4250	£ 255
Repairs	350
Wages, 1½d. per M	400
Coke, 40.7 lb. per M at £1 0s. 4d.	2608
	£3613
	Approximately $17,340.00

STEAM IN RETORTS

6 per cent. interest and depreciation on 1.7 settings at £2000 per setting		£ 200
Repairs at £150 per setting		255
Wages for 1.7 settings		85
Coke as furnace fuel per M	15.6 lb.	
Coke in retorts per M	14.7 lb.	
Total coke per M	30.3 lb.	
Total coke used, 1907 tons at £1 0s. 4d.		1940
		£2480
		Approximately $11,900.00

Saving per M by admission of steam to retorts 1.9d., about 3.8 cts.

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[1] London Jnl. of Gas Ltg., 1908, page 441.

[2] London Jnl. of Gas Ltg., 1909, page 296.

Koerting and Geipert[1] further state that by a heat balance of retort reactions, it is shown that the greater portion of the heat entering the retort is utilized in heating the coal, or coke, thereby creating a considerable stock of heat, and consequently the conditions for producing water gas in the retorts are especially favorable. If the coke should be cooled down from 1250°C. to 1000°C. (2282°F. to 1832°F.), 7141 calories would be evolved in the 63.76 kilos of coke per 100 kilos of coal (12,886 b.t.u. per 63.76 lb. of coke per 100 lb. of coal); in the water-gas reactions, the heat absorbed per cubic meter of steam amounts to either 1800 or 981 calories (202 or 110 b.t.u. per cu. ft. of steam), according to whether the carbon is oxidized to CO or to CO_2 respectively, and 3.97 or 7.28 cubic meters of steam can be transformed to carbonic oxide, or acid, and hydrogen (64 or 117 cu. ft. per 100 lbs. of coal).

During the test made at Mariendorf, Germany, the production of water gas per 100 kilos of coal amounted to 6.3 cubic meters, 101 cu. ft. per 100 lb. of coal and consequently this production fell far below the theoretical figures. These tests showed that the coke consumption was 14.5 per cent. without steam, and but 14.1 per cent. with steam, and it was found necessary during these tests to increase the furnace draft during the period when no steam was admitted into the retort.

These authorities also claim a further advantage in admitting steam into the retort in that the requirements of carbon for the production of water gas are met in part by the graphite formed in the retorts, thereby utilizing what is ordinarily considered a source of trouble and expense; they calculate that about 18 per cent. of the carbon required for the 6.3 cubic meters of water gas produced per 100 kilos of coal at Mariendorf, was supplied by this graphite.

Another advantage claimed by Koerting and Geipert for this steaming process lies in the fact that at least 10 per cent. more gas, as explained above, can be made per bench per day without any additional wages or cost of repairs. Moreover, to the cost of the water-gas generator for the dry system must be added the cost of a building and of a relief holder, amounting to $40,000 in all, while the steaming process simply requires the installation of one extra bench for each 14, this cost amounting to $12,000. The repairs allowed for the water gas plant would average $1680 a year, while those of a vertical bench would be about $720.

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[1] London Jnl. of Gas Ltg., 1908, pages 400 and 404.

The cost of producing 1000 cubic meters (35,300 cu. ft.) of water gas in an independent plant would then be:

652 kilos (1434 lb.) of coke at $4.80 per 1000 kilos (2200 lb.)	$3.12
Wages	0.48
Total	$3.60

while the production of the water gas in the retort would entail only the cost of the fuel, which in this case would be 207 kilos (455 lb.) of carbon, or 235 kg. (517 lb.) of coke, amounting to $1.13, or a saving of $2.47; the total cost of the two systems, including interest, depreciation, and repairs would be about $4.32 per thousand cubic meters (12.2¢ per M. cu. ft.) of water gas produced by an independent generator, and $2.79 (7.9¢ per M. cu. ft.) by admitting steam into the retorts.

This statement is open to criticism, due to the fact that it calls for less coal being used when steam was admitted into the retorts, although the charges were kept in 1½ hours longer, as well as to the paradoxical statement that, although the temperature of the coke was reduced from 1250°C. to 1000°C. (2282°F. to 1832°F.) by the admission of steam, the retort was in better condition for carbonizing a fresh charge of coal. It may also be said that if the water gas was really produced by the heat stored up in the coke, it should not be necessary to keep the charge in the retort for 1½ hours for the mere purpose of utilizing that heat in the production of water gas, as a more rapid flow of steam would more quickly bring down the temperature of the coke, and if no extraneous heat were required for the production of water gas, the same quantity could have been made more quickly and without any disadvantage to the working conditions.

Dr. Strache[1] in criticising the statement of Koerting and Geipert claims that in an independent water-gas plant the maximum production per square foot of grate area per run amounts to 139.4 cu. ft. of gas before the temperature of the surface of the coke falls below the economical dissociation point; under these conditions, with 1.29 sq. ft. of retort area, and with the production of 2260 cu. ft. of water gas claimed, Strache points out that the water gas must have been produced under very uneconomical conditions, and that the quantity of steam admitted into the retort must have been from ten to twenty times the theoretical amount; also, on the basis of the lesser amount of steam used, the boiler coal required to generate this quantity will be 37 lb. per 1000 cu. ft. of water gas. STRACHE finally points out that the cost of the water-gas plant assumed by KOERTING and GEIPERT, as well as the cost of repairs thereon, applies to a plant for the manufacture of carburetted water gas, which is not a proper assumption.

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[1] London Jnl. of Gas Ltg., 1909, page 95.

KOERTING and GEIPERT in defence of their statement claim that the steam used was accurately measured, and was found to be 35.4 lb. per 1000 cu. ft. of water gas made, or less than what would be used in an ordinary water-gas plant. In further defending the admission of steam into the retort, they state that the cross-section of the retort at its hottest point is double that assumed by STRACHE; that the height of the column of coke is several times that of the fuel bed in a water-gas generator; and, finally, that heat is being constantly conveyed from the exterior into the interior of the retorts while water gas is being made.

TERHAERST and TRAUTWEIN[1] point out that when steam is admitted into the retort, the charge is worked off in 12 hours, without steam in 10½ hours, and that the heat required during the additional 1½ hours for the manufacture of water gas, radiation, conduction, etc., cannot be supplied by the sensible heat of the coke, especially as the setting is maintained under fire during this period; therefore, the expenditure of fuel when steam is admitted must be relatively greater than when it is not admitted.

It is very important to remember that when mixed gas is being made in the retort, both the carbonizing period and the gas production are increased by about 10 to 12 per cent., but we must not forget that the retorts are not available for coal-gas manufacture during this period; this condition requires a greater number of retorts to produce mixed gas than would be the case if straight coal gas were made, this fact increasing the working charges, depreciation, repairs, etc., and the scurf consumed in the production of water gas must also be charged against same, as this carbon has a selling value. Of this basis TERHAERST and TRAUTWEIN worked out the following costs for water-gas production:

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[1] London Jnl. of Gas Ltg., 1909, page 903.

In independent generators:

36.7 lb. of coke at £1 0s. 4d	4.0d.
Wages	0.3d.
Total	4.3d.
	About 8.5¢

In vertical retorts:

14.5 lb. of coke at £1 0s. 4d	1.6d.
2.8 lb. of scurf at £2 10s. 10d	0.7d.
Total	2.3d.
	About 4.5¢

Assuming that the plant operates 300 days per year, producing 140 million cu. ft. of water gas during this period, thus requiring 15.9 settings instead of 14, the total cost would be:

In independent generators:

Interest and depreciation on £4250 at 6 per cent	£ 255
Repairs	75
Mfg. charges at 4.3d. as above	2508
Total	£2838
Approximately	$13,622.00

In vertical retorts:

Interest and depreciation on £2500 × 1.9 at 6 per cent	£ 285
Repairs for 1.9 settings	285
Wages	95
Mfg. charges at 2.3d. as above	1341
Fuel consumption for 1.9 settings	1159
Total	£3165
Approximately	$15,192.00

or a difference of £327 ($1570.00) in favor of the independent water-gas generator.

DESSAU VERTICALS AT PROVIDENCE, R. I.[1]

This installation consists of two stacks, each containing six benches of ten retorts, in use, and a third stack which had not been fired. The retorts are each 13 ft. 2 in. long, 22½ in. by 9 in. at the top, and 27 in. by 14 in. at the bottom, the retort walls being 3 in. thick at the top, increasing to 4 in. at the bottom.

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[1] C. D. Miller, Proc. A. G. I., 1911, page 493.

The retorts are arranged in two rows of five each, the producer being located at the side, lengthwise of the bench. The passages are so arranged that the producer gas and air enter the setting at six places, four between the retorts and one at either end of the benches. The grate measures 3 ft. 4 in. by 4 ft., the total depth from the grate bars to the charging door being 18 ft., but, as the producer gas is taken off at a point about 3 ft. above the grate, that figure necessarily represents the active depth of fuel when the fire is clean, the remainder of the height serving as fuel storage. Combustion takes place at the bottom of the retorts, the resultant products making four horizontal passes, alternately away from and toward the producer, leaving at the top of the setting in a down direction through the recuperators and then under a steam generator, passing from thence to the main flue, the generator supplying steam to the fire.

Six months' operation of these two stacks gave the following results:

Time of charge	9 to 11 hours
Number of retorts charged	57,983
Average weight of charge	990 lb.
Gas per pound of coal	4.94 cu. ft.
Candle power, Bray No. 7 flat flame	13.21
Candle feet	65.26
Calorific value	622 B.t.u.
Bench fuel per 100 lb. of coal	19.26 lb.
Yield per 2240 lb. of coal	11,072 cu. ft.
Tar per 2240 lb. of coal	15.6 gal.
Ammonia per 2240 lb. of coal	6.47 lb.
Coke made	70 per cent.
Coal handled per man per day	12.045 tons

Mr. Miller further stated that carbon formed in the retorts very rapidly, particularly in the lower portions, and that the average time of scurfing did not fall much below 40 hours per retort, each retort being scurfed every 20 days, producing a loss in capacity of over 8 per cent. In addition to this, the amount of coal charged decreased as the carbon deposit increased, producing an additional loss of about 1.6 per cent. The pressure of the gas at the bottom of the retort amounted to 10 to 12 in. of water during the first period of the charge, decreasing to 3 to 4 in. during the last few hours. The charges hung in the retort at times, this being probably due to the low temperature at the extreme bottom of the retort; the amount of coke burned in the producers per square foot of grate area per hour varied from 14 to 16 lb.

DESSAU VERTICALS AT MARIENDORF, GERMANY

E. Koerting[1] reports that the setting at Mariendorf differs from the ordinary Dessau type in that it contains eighteen retorts instead of twelve, the required ground area being the same in both instances, but the retorts in the new setting being of smaller cross-section and arranged in three rows of six instead of two rows. The coal carbonized contained 2.91 per cent. of moisture on drying at 110°C. (230°F.), 4.28 per cent. of ash, and 92.81 per cent. of combustible, the ultimate analysis being 60.82 per cent. of carbon and 31.99 per cent. of volatile matter, the resultant coke being of good quality and containing 6.59 per cent. of ash.

The results given below were obtained during a test run of 105 hours, these results being compared with those secured in May, 1908, with a twelve-retort setting, the gas being corrected to 60°F. and 30-in. barometer.

	18 retorts	12 retorts
Coal carbonized per 24 hours, lb.	43,792	30,800
Gas made per 24 hours, cu. ft.	262,898	190,183
Coke used as fuel, lb.	5,185	4,341
Yield per 2240 lb.	13,430	13,815
Gross calorific value, B.t.u.	542.6	536.7
Candle power, No. 2 Met. burner	10.0	....

The charge during this test run was 10½ hours on the setting of 18's, and steam was admitted for 2 hours.

DESSAU VERTICALS AT COLOGNE, GERMANY

This plant consists of twenty-four benches (1908) of ten retorts each, the retorts being 4 meters (13.12 ft.) long. Ruhr coal is carbonized, the charges being burned off in 11 hours, steam being admitted into the retort during the last 2 hours, the entire installation of 240 retorts requiring eighteen men per 24 hours, with two extra men for cleaning ascension pipes and hydraulic main, as well as two men on the coke conveyors, or twenty-two men total, the gas make per man being 154,000 cu. ft. The furnaces are equipped with water-cooled grates to facilitate clinkering, which requires about 50 minutes every 32 hours. Scurf is removed every 30 to 35 days, requiring 11 hours in burning off; thus only one charge is missed during this period.

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[1] London Jnl. of Gas Ltg., 1910, page 27.

The results given by test runs under different conditions, and with and without steam, are given below:

Length of test in hours	50	50	44
Admission of steam	None	8th to 10th hour	9th to 11th hour
Charge worked off in hours	10	10	11
Total coal used, tons	25.35	25.95	20.40
Coke for fuel, per cent. of coal carbonized	15.60	17.28	17.23
Coke for fuel, lb. per M	29.90	29.01	28.08
Per cent. CO_2 in flue gas	18.70	18.48	18.36
Yield per ton, 60° and 30 in	11,931	13,069	13,618
Specific gravity of gas	0.44	0.45	0.45
Candle power[1]	12.35	9.47	8.20
Calorific value per foot	575.40	553.10	547.30
Dry coke obtained, per cent. of coal	72.74	71.30	71.87
Tar and liquor, per cent. of coal	21.62	17.30	14.04
Composition of Gas by Volume:			
Carbonic acid	1.10	2.3	2.0
Heavy hydrocarbons	3.50	3.0	2.6
Carbonic oxide	8.90	12.0	13.0
Methane	28.74	26.2	22.9
Hydrogen	53.73	52.5	56.1
Nitrogen	4.03	4.0	3.4
Analysis of the Coal Used:			
Test number	1 and 2	3.00	4.00
Coal proper, per cent. by weight	85.78	88.32	84.03
Ash	11.32	9.23	13.24
Moisture	2.90	2.45	2.73
Gas	28.45	27.34	30.74
Coke	71.55	72.66	69.26
Average Composition of Coal Proper:			
Carbon	85.32		
Hydrogen	5.45		
Oxygen	8.02		
Nitrogen	1.31		
Sulphur	0.90		
B.t.u. per pound	13,025		

Average Composition of Cold Coke Used for Heating Retorts:			
Test number	1 and 2	3.00	4.00
Combustible	74.87	73.26	75.58
Ash (on moist coke)	13.01	15.52	11.52
Moisture	12.12	11.22	12.90
Elementary Analysis of Coke Used:			
Carbon	74.17		
Hydrogen	0.64		
Oxygen	0.46		
Nitrogen	0.86		
Sulphur	0.79		
Ash	10.49		
Moisture	12.59		
B.t.u. dry per pound	12,622		
B.t.u. moist per pound	11,084		

[1] Illuminating power given in candles per cubic foot, measured in an ELSTER burner at the rate of 5.3 cu. ft.

Tar yield was an average of 4.16 per cent. by weight of the coal carbonized.

Composition of tar	From hyd. main	From condensers
Specific gravity	1.143	1.095
Water, per cent	3.000	1.500
Light oil (to 170°C. = 338°F.), per cent	2.000	5.300
Middle oil (170°C. to 230°C. = 446°F.), per cent	11.000	20.000
Heavy oil (230°C. to 270°C. = 518°F.), per cent	12.500	11.300
Anthracene oil (270°C. to 380°C. = 716°F.), per cent	29.500	30.700
Pitch, per cent	42.000	31.200

The temperatures during test run averaged something over 1300°C. (2372°F.) in the lower combustion chamber; 80° to 90°C. (176° to 194°F.) less at the middle of the height of the retorts; and about 550°C. (1022°F.) for the waste gases.

The ammonia yield amounted to 0.256 per cent. of the weight of coal carbonized, or about 22 lb. of sulphate per ton, while the sulphur in the gas averaged 146 grains per 100 cu. ft. The cyanogen in the gas at the inlet to the condenser amounted to 79.8 grains per 100 cu. ft.

MR. THOMAS HOLGATE gives the following very interesting results obtained with DESSAU verticals:[1]

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[1] London Jnl. Gas Ltg., 1908, Sept. 8, page 640; Sept. 15, page 707; Sept. 22, page 785; Sept. 29, page 840; Oct. 13, page 117.

TABLE XXXIII

Steam	DESSAU		ZURICH		MARIENDORF	
	None	Slight	None	Copious	None	Copio
Length of retort, ft.	13.12	13.12	13.12	13.12	16.4	16.4
Coal per day, tons	1.161	1.161	1.182	1.119	1.277	1.14
Temp. of settings °C., base			1,330	1,378	1,288	1,30
Temp. of settings °F., base			2,426	2,436	2,350	2,37
Temp. of settings °C., higher level			1,204	1,243	1,209	1,22
Temp. of settings °F., higher level			2,200	2,270	2,210	2,24
Coal used	New Pelton		Saar		Silesian	
Yield per ton, cu. ft.	11,259	11,680	12,180	13,820	11,400	13,81
C. P. per 5 cu. ft.	16.45	15.37	9.11	6.23	15.9	12.1
Cal. value, B.t.u., gross	604	596	591	561	585	53
Cal. value, B.t.u., net	543	535	527	500	523	48
Coke produced, per cent			69.3	67.2	64.04	63.7
Coke fired, per cent	16.25	16.25	13.9	15.0	14.60	14.1
Matter volatilized, per cent			30.7	32.8	35.96	36.2
Gross B.t.u. per pound of matter volatilized per ton			10,463	10,548	8,274	9,14

TABLE XXXIV.—COMPARISON OF EFFECT OF STEAMING

	DESSAU	ZURICH	MARIENDO
1. Decrease in candles per ton	1,138.0	4,972.0	2,666.0
2. Decrease in candles per ton, per cent	3.0	22.4	7.3
3. Increase in gross B.t.u. per ton	160,844.0	554,640.0	749,970.0
4. Increase in gross B.t.u. per ton, per cent	2.3	7.7	11.2
5. Increase in net B.t.u. per ton	135,163.0	496,012.0	677,472.0
6. Increase in difference between gross and net B.t.u.	25,681.0	58,628.0	69,498.0
7. Ratio of line 3 to line 6	6.26	9.46	10.75
8. Increase in gross B.t.u. per pound of matter volatilized per ton		85.0	870.0
9. Increase in gross B.t.u. per pound of matter volatilized per ton, per cent		0.8	10.5

HOLGATE states that the first significant fact to be noticed i that the 5-meter (16.4 ft.) retorts while having a length amounting to 25 per cent. more than the others, distil only 10 per cen more coal, and that the advantages here appear to lie in permittin a lower oven temperature with a greater suitability for makin water gas. The table shows that the 4-meter (13.12 ft.) retort were operated at a higher temperature than the 5-meter, this fac probably accounting for the higher calorific and lower illuminatin value in the former, although in regard to the candle power, i must be remembered that this was measured with a CARPENTE at Mariendorf, and with a flat-flame burner at Zuric in this connection the higher working temperature and the low percentage of volatile matter in the saar coal must be taken into account to understand the low candle power recorded, coincident with the high calorific power per ton.

It is in the supplementary manufacture of water gas that we find the 5-meter retorts appearing with most credit, and Table XXXIV shows that the advantage of the increased yield is not lost when the calorific value of the gas per ton of coal is taken into account. The admission of steam reduces the lighting value per ton of coal by 3 per cent. with slight steaming at Dessau, by 7.3 per cent. with copious steaming at Mariendorf, and by 22.4 per cent. with copious steaming at Zurich. The tables show the relation that exists in the hydrogen content of the gas produced, this being shown by the difference between the gross and the net calorific value, line 7, Table XXXIV, giving a ratio which appears to furnish a good index of the efficiency of the retort as a water-gas producer; the ideal ratio would be 12.2, and this is very nearly reached in the settings at Mariendorf.

Holgate arrives at this 12.2 ratio in the following manner: Ideal water-gas reactions form 28 parts by weight of CO and 2 of H, with a total calorific value of 136,030 calories, which produce on combustion 18 parts of water by weight. The latent and sensible heat of the water vapor thus formed would amount to 11,136 calories, which figure represents the difference between the gross and the net calorific value of 2 grams of hydrogen; consequently if the actual amount of the latter yielded by any gas is known, it is possible to tell how much hydrogen the gas contained. Therefore the 11,136 calories correspond to 68,360 calories of heat developed by hydrogen, and with true water gas this heat should be accompanied by 67,670 calories from the CO; thus 136,030 ÷ 11,136 = 12.2, and by this rule three-quarters of the water gas was converted to carbonic oxide at Mariendorf, four-ninths at Zurich, and but 2 per cent. at Dessau. The superiority of the 5-meter retort for producing water gas is confirmed by the last line in Table XXXIII and lines 8 and 9 in Table XXXIV, where the increase in gross calorific power of the gas produced by Mariendorf is 870 B.t.u., and only 85 B.t.u. at Zurich.

As regards Table XXXIV, it may be stated that if the increase in production for the steaming and non-steaming periods be adjusted according to Holgate's calculations, the figures in lines

3 and 5, or the increase in gross and net B.t.u.'s per ton, would become 212,348 and 181,421 instead of 160,844 and 135,163; both of these figures are based on two assumptions, but if the calculations should be made upon only one of these assumptions, or upon the inference of the quality and quantity of the water gas from the average analysis of the purified gases, these figures would become 174,048 and 156,063 respectively, or in other words, we should have 507 cu. ft. of water gas produced and this in turn would yield 174,048 and 156,063 B.t.u.'s, respectively gross and net. These figures closely approximate the average of the two preceding ones, and after considering the magnitude of the respective amounts, the differences being only about one-fortieth of the totals, as well as the existing difficulties as to data, we find that the various computations are not very far apart.

Table XXXIII shows that the candles per ton at Dessau and at Mariendorf without admitting steam into the retort are 37,042 and 36,252 respectively, a difference of only 2 per cent.; KOERTING gives the illuminating power as 17.7 and 10.8 candles respectively, but this disparity may be explained by referring to Table XXXV in which the results of the January and May tests are given.

TABLE XXXV

Date	Gross calorific value			Illuminating value		Detail
	Cu. ft. per ton	Per cu. ft.	Per ton of coal	Per 5 cu. ft.	Per ton of coal	
January	13,478	546	7,358,988	10.80	29,112	Mariendorf retorts, steamed.
May	13,810	537	7,415,970	12.16	33,586	
Difference	+332	−9	+56,982	+1.36	+4,474	
January	11,292	597	6,741,324	17.70	39,973	Dessau retorts, not steamed.
May	11,259	607	6,800,436	16.45	37,042	
Difference	−33	+7	+59,112	−1.25	−2,931	
May	11,400	585	6,669,000	15.90	36,252	Mariendorf, not steamed.

It will be noted that the agreement in gross calorific power is striking, each pair being concordant and exhibiting an improvement in 5 months amounting to almost 60,000 B.t.u.'s, but the lighting results exhibit a great contrast to these. The non-steaming tests during May show a difference of 790 candles in favor of Dessau, consequently leaving 3456 − 790 = 2666 candles to be accounted for, and Holgate examines these results to determine whether any part of this difference is due to the decomposition of hydrocarbons by an excess of steam. He states first that Table XXXV shows that the non-steaming working in the two plants is almost identical as to candles and B.t.u.'s per ton, and that the 7,358,988 − 6,741,324 = 617,664 B.t.u.'s per ton of coal, corrected for the difference in the two coals used, or by 6,800,436 − 6,669,000 = 131,436, will give 749,100 B.t.u.'s credit to the water gas made in the 5-meter retort; furthermore, water gas, when freed from incombustibles, will have 342 B.t.u.'s per cubic foot; therefore the increase in make, or 13,478 − 11,292 = 2186 cu. ft. × 342, or a total of 747,612 B.t.u.'s, should be equal to the calorific value of the water gas made; this figure seems to prove that at Mariendorf one-tenth of the heating power of the gas is due to steaming, and therefore to the resulting water gas.

Table XXXVI shows the effect of the introduction of steam

TABLE XXXVI

	Dessau	Zurich	Mariendorf	Line
Increased vol. observed per ton, cu. ft.	421	1640	2410	1
Increased vol, per cent.	3.74	13.46	21.14	2
Increased gross cal. value, per cent.	2.30	7.70	11.20	3
Ratio lines 2 to 3	0.61	0.58	0.53	4
Ratio gross cal. value of water gas per cu. ft. to that of undiluted coal gas	0.568	0.58	0.586	5
Do., calculated upon standard basis.	0.57	0.58	0.598	6
Corrected volume per ton required by standard basis, cu. ft.	461	1617	2178	7
Corrected increased volume per cent. required by standard basis, in lieu of line 2	4.11	13.25	18.72	8

The standard referred to by Holgate is based on an ideal gross calorific value of water gas.

Holgate further states that his studies of this subject indicate that the admission of steam into the retort is without effect upon the results of dry distillation, except that it adds a quantity of water gas to the coal gas already formed.

He assumes ethylene and benzene to be the unsaturated hydrocarbons, and calculated Table XXXVII on this basis for the **WOODALL-DUCKHAM** continuous and the **DESSAU** intermittent retorts, using figures given by **MESSRS. BROADBERRY** and **COLEMAN** calculated to a 100 per cent. basis, and others received from **MR. H. W. WOODALL.**

TABLE XXXVII.—CALORIFIC VALUE PER TON OF COAL AT NINE ELMS GAS WORKS (STEAMED)

	Volume, cu. ft.	Calorific value			Line
		Gross	Net	Difference	
CO	920.54	314,824	314,824	....	1
H_2	5,695.94	1,959,403	1,640,431	318,972	2
CH_4	4,354.26	4,659,058	4,171,381	487,677	3
Simple combustibles	10,970.74	6,933,285	6,126,636	806,649	4
Complex combustibles	357.78	478,277	466,250	12,027	5
Total combustibles	11,328.53	7,411,562	6,592,886	818,677	6
Non-combustibles	1,094.47	....	....	....	7
Observed values	11,423.00	7,411,562	6,592,886	818,676	8

TABLE XXXVIII.—CALORIFIC VALUE PER TON OF COAL AT DESSAU GAS WORKS, SECOND DAY (STEAMED)

	Volume, cu. ft.	Calorific value			Line
		Gross	Net	Difference	
CO	804.5	275,139	275,139	....	1
H_2	6,753.4	2,323,169	1,944,979	378,190	2
CH_4	3,317.3	3,549,511	3,177,973	371,538	3
Simple combustibles	10,875.2	6,147,819	5,398,091	749,728	4
Complex combustibles	348.1	809,568	846,816	37,248	5
Total combustibles	11,223.3	6,957,387	6,244,907	712,480	6
Non-combustibles	456.7	....	....	....	7
Observed values	11,680.0	6,957,387	6,244,907	712,480	8

An examination of these two tables shows that using English Coal, the **WOODALL-DUCKHAM** retorts at Nine Elms produce 105.23 cu. ft. of combustible gas per ton of coal more than the **DESSAU**, and 637.77 cu. ft. more of incombustible gas. **MR. HOLGATE** also states that the amount of methane (CH_4) at the Nine Elms works, as compared with the **DESSAU** plant, is a notable feature, but figures received from the South Metropolitan works, London, show that in this particular, horizontal retorts exceed either of the above.

A comparison of calorific value and candle power, for various retorts, is shown in Table XXXIX.

TABLE XXXIX

	Not steamed		Steamed	
	Gr. B.t.u. per ton	Candles per ton	Gr. B.t.u. per ton	Candles per ton
Mariendorf vertical	6,669,000	36,252	7,415,970	33,586
Dessau vertical	6,799,310	37,049	6,957,387	35,904
Zurich vertical	7,198,380	22,192	7,753,020	17,220
Poole vertical	7,807,748	40,398	....	....
Nine Elms vertical	....	....	7,411,562	39,306
Nottingham horizontal	7,734,881	37,460	....	....

The burners used in the above candle-power tests were the Metropolitan No. 2 at Mariendorf, Dessau, Poole, and Nine Elms; a flat flame at Zurich; and a London No. 1 at Nottingham. If a proper allowance is made for the difference in test burners used, on the basis of 15.6 with the London Argand to 17 with the No. 2 Metropolitan, the candles per ton amounting to 37,049, with the verticals at Dessau, become 40,820 candles per ton.

Table XXXVII plainly shows the superiority of the Woodall-Duckham retort at Nine Elms in the matter of methane content, the comparison being made because the amount of water gas was probably not very divergent.

TABLE XL.—CALORIFIC VALUE PER TON OF COAL, BRIGHT ORANGE HEAT, NOTTINGHAM HORIZONTALS

	Volume, cu. ft.	Calorific value			Line
		Gross	Net	Difference	
CO	1,676.0	573,192	573,192	....	1
H_2	5,765.3	1,983,263	1,660,406	322,857	2
CH_4	3,685.8	3,943,806	3,530,996	412,810	3
Simple combustibles	11,127.1	6,500,261	5,764,594	735,667	4
Complex combustibles	541.5	1,234,620	1,146,897	87,723	5
Total combustibles	11,668.6	7,734,881	6,911,491	823,390	6
Non-combustibles	337.4	....	....	....	7
Observed values	12,006.0	7,734,881	6,911,491	823,390	8

In making a comparison with horizontal retorts, the figures given in Table XL should be employed for non-steaming operations.

Table XLI gives the calorific value, not steamed, for the DESSAU verticals.

TABLE XLI.—CALORIFIC VALUE, DESSAU RETORTS, FIRST AND THIRD DAYS NOT STEAMED

	Volume, cu. ft.	Calorific value			Line
		Gross	Net	Difference	
CO	633.88	216,787	216,787	....	1
H_2	6,465.14	2,224,008	1,861,960	362,048	2
CH_4	3,376.01	3,612,331	3,234,218	378,113	3
Simple combustibles	10,475.03	6,053,126	5,312,965	740,161	4
Complex combustibles	326.85	746,184	788,850	42,666	5
Total combustibles	10,801.88	6,799,310	6,101,815	697,495	6
Non-combustibles	457.12	....	....	....	7
Observed values	11,259.00	6,799,310	6,101,815	697,495	8

HOLGATE concludes that the advantages of vertical retorts are not inherent in any radical superiority so far attained over distillation in horizontal retorts at high temperatures, but that this does not mean that the opportunity for improvement does not exist, but rather that by a continuous method horizontal retort working, as hitherto practised, may be eclipsed.

The Woodall-Duckham Vertical Retort.—The WOODALL-DUCKHAM vertical retort is one of continuous operation, and each particle of coal during its passage through the retort undergoes exactly the same treatment. The descending charge of coal in the retort gradually becomes heated, the gases and by-products being thereby evolved as the coal reaches the critical temperature governing the liberation of the particular constituent. The gases, in passing to the top of the retort, are never compelled to pass through a hotter mass of material than that from which they were evolved, thus making the process one of fractional distillation.

The gases evolved in many vertical retorts are poor in quality, the resultant tars being of a low specific gravity and containing a high percentage of light oils. These same light oils are usually broken up in horizontal or inclined retorts, due to their passing through the free space above the charge, thus forming carbon in the retort, lamp black and pitch in the ascension pipes, heavy tar in the hydraulic mains, as well as naphthalene and sulphur in the gas. Due to the division at the top of the retort and the sliding plate behind it, a space is provided in the WOODALL-DUCKHAM retort the size of which can be so regulated as to obtain the most efficient results from the cracking up of the tarry vapors. Due to this method of operation, the gases evolved are not indiscriminately subjected to the action of heat from the walls of the retort, as would be the case in inclined or horizontal retorts. This device also permits of securing candle powers as high or even higher than those usually obtained in horizontal retorts, this high candle-power production being greatly assisted by the high heats existing around the top of the retort. The construction of the retort and its appurtenances is such that the speed of the charge through the retort, as well as the temperature of carbonization, can be readily adjusted to obtain the maximum conditions for varying coals.

The construction of the retort is very simple, the vertical flues being so arranged as to make it possible to see the heats and the condition of the whole retort from the top platforms. The size of the retort, as well as the brick walls, which are only 4½ in. thick, make it possible for the retort to be easily repaired, a man being able to work inside and to use ordinary bricks. All joints are tongued and grooved, the multiplicity of these joints acting to prevent the cracking which so often takes place in fire-clay retorts made in one piece.

The work of the attendant at the top of the retorts is limited to the opening of the coal valve every 3 hours and the filling of the auxiliary hopper with coal, this operation requiring about 10 seconds per retort, after which he need only watch the coal feed indicators. The attendant on the lower floor discharges the coke every 3 hours by turning the handle on the water-sealed door, this requiring about 5 seconds per retort. The producer charging door is located on the same level as the top of the retorts, and the attendant charges the producer by means of a chute connected with a coke hopper.

The retorts, Fig. 40, are 25 ft. long and have a tapering cross-section, the walls of the retorts being formed of tongued and grooved brick, for the prevention of leakage at the joints, so panelled out at the back as to permit the heat to readily pass through the walls to the coal charge in the retort, thus also presenting a large surface to the heating gases in the flues. The heating flues are arranged vertically, the division walls, Fig. 41,

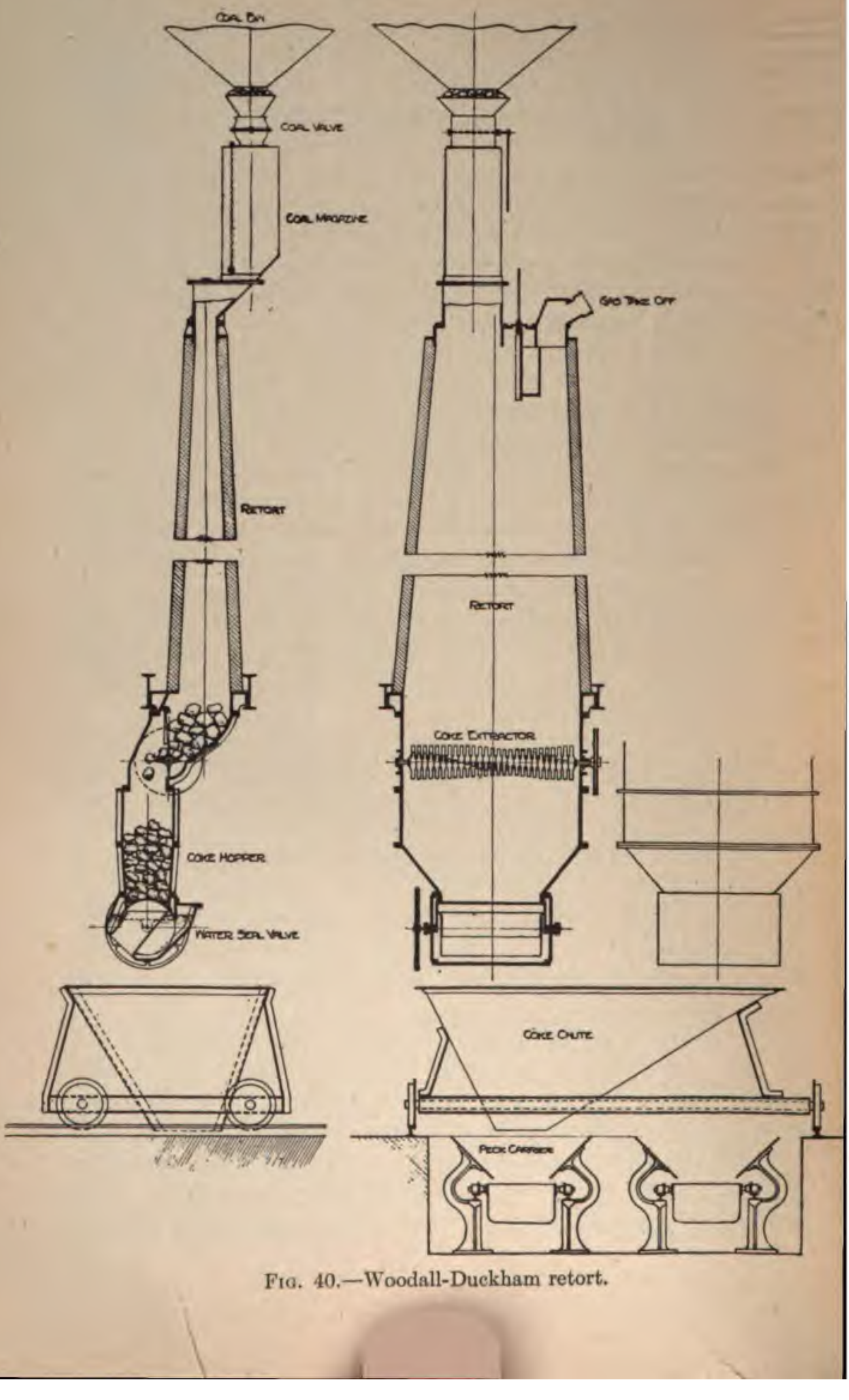

FIG. 40.—Woodall-Duckham retort.

of tongued and grooved brick being bonded into the back walls of the flues and into the walls of the retort, in this manner forming a very strong and simple construction. The gas from the producer, as well as the secondary air from the recuperator are admitted at the top of the vertical flues and burn downward.

One producer and recuperator are provided for each bench, the producer being of large dimensions and so constructed as to maintain a constant depth of fuel from the grate bars to the gas take-off. The walls between the waste gas and the secondary air flues are constructed of the same tongued and grooved panelled blocks provided in the retort walls; dampers are provided at a point near the bottom of the retort, so arranged that the draft can readily be regulated on each side of every retort, while main dampers are provided on the waste gas take-off from the recuperator.

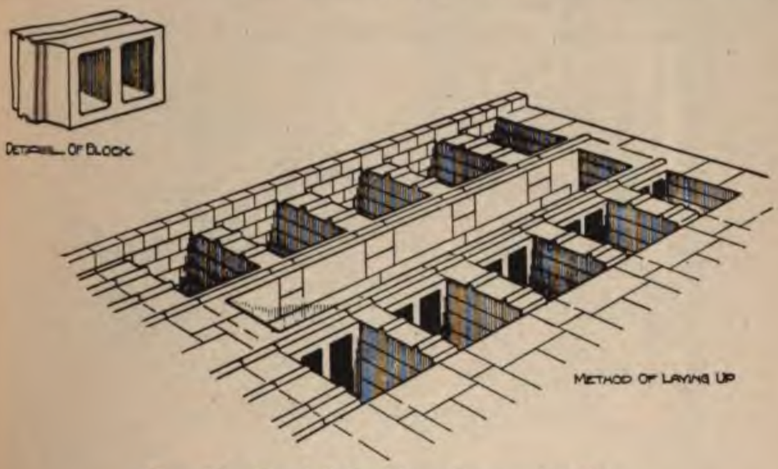

FIG. 41.—Woodall-Duckham flue construction.

The bottom of each retort is provided with a device for regulating the coke discharge, Fig. 40, the efficiency of the entire setting being dependent upon the proper operation of this discharge arrangement; the design of this device is such that no portion of it is compelled to shear through the coke, and it is free to operate without being compelled to bear the weight of the charge above. The driving mechanism is so arranged as to permit of easily varying the speed of discharge from each retort to suit the temperature of the setting and the quality of the coal being carbonized. This extractor is placed at the bottom of a curved plate, the latter forming the back of a cast-iron hopper attached to the bottom of the retort, this plate taking the weight of the charge in the retort; the curved plate is provided with a manhole and inspection holes, to permit of easy access to the retort. Hangers are suspended in the hopper from a transverse shaft located above the extractor, these hangers acting to prevent the passage of any coke over the extractor unless regulated by the action of the latter. The extractor roller is constructed of a series of cast-iron arms mounted on a square shaft, the arms being cast in the form of a cross, each cross being so located on the shaft as to give it a slight lead over its neighbor, the entire roller being thus formed of helical blades.

This construction ensures an even discharge of coke throughout the whole revolution of the roller, thus regulating an even travel of the charge through the retort by causing a gradual descent of the coke.

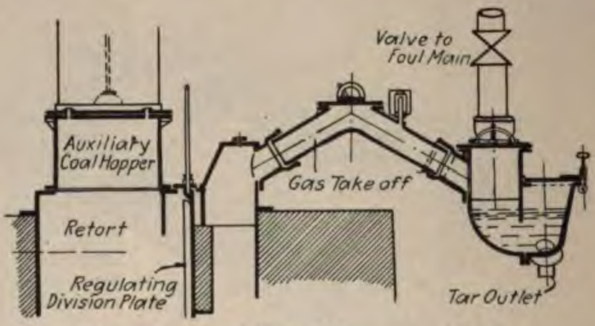

FIG. 42.—Woodall-Duckham gas take-off.

The gas take-off, Fig. 42, is so arranged that it can be readily cleaned, while the hydraulic main is of quite simple construction.

The entire setting, Fig. 43, including the producers, recuperators, and the various attachments and working parts, are supported on a structural steel floor above the ground. The bench buckstays are attached to the floor beams and are fixed on both sides of the settings at their centers, being also connected across the top of the bench. The coal and coke hoppers are continuous above the settings, being supported on columns built up from the connecting beams, these hoppers being provided with chutes for feeding the producer and for filling the retorts.

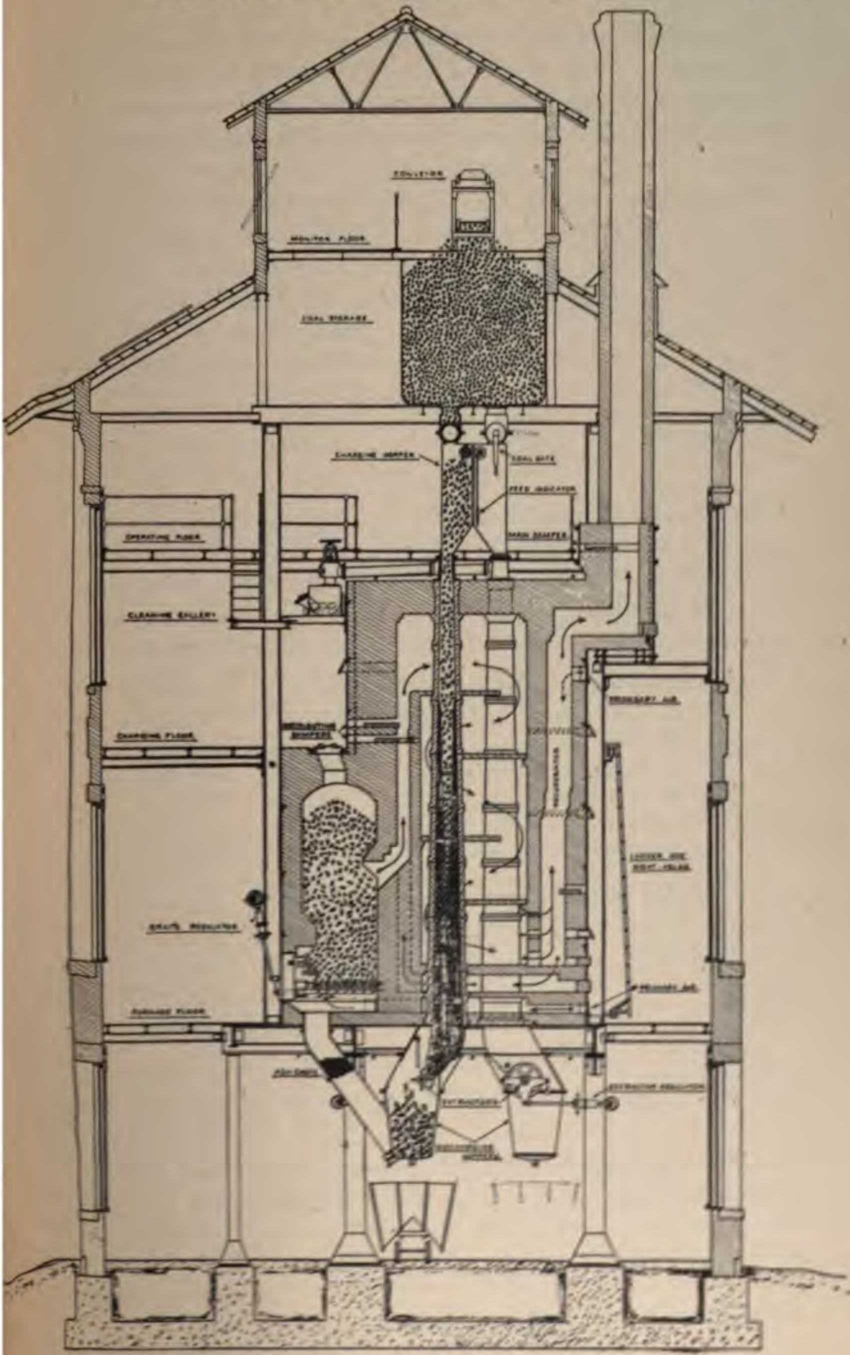

FIG. 43.—Woodall-Duckham vertical setting.

A series of temperature determinations in the center of WOODALL-DUCKHAM vertical retorts made by MESSRS. R. Ross and J. P. LEATHER,[1] Burnley, England, are given below, the tests having been made with a 25-ft. pyrometer. The average results were:

Depth below rodding holes	Temp., °F.	Temp., °C.	Depth below rodding holes	Temp., °F.	Temp., °C.
21 ft. 4 in.	1636	890	11 ft. 6 in.	1607	875
19 ft. 4 in.	1640	893	9 ft. 6 in.	1456	824
17 ft. 4 in.	1661	905	7 ft. 6 in.	1319	715
15 ft. 4 in.	1233	667	5 ft. 6 in.	968	520
16 ft. 8 in.	1112	600	3 ft. 6 in.	601	316
15 ft. 6 in.	1373	745	2 ft. 6 in.	568	298
13 ft. 6 in.	1355	735			

Woodall-Duckham Verticals at Derby, Conn.—In 1912 a setting of four benches of four retorts each was built at Derby, Conn., each retort being 24 ft. 8 in. long, with a cross-section of 30 by 20 in. at the bottom and 24 by 9 in. at the top. An 8-day test made on this installation gave the following results:

Yield per pound of coal	5.42 cu. ft.
Candle power, Sugg D burner	15.57
Candle feet	84.40
B.t.u., Junker's calorimeter	615.00
B.t.u., by analysis	645.00
Ammonia, per gross ton of coal, lb.	5.88
Tar, per gross ton of coal, gal	12.66

The composition of the gas was:

Benzol	0.55	Carbonic oxide	6.90
Carbonic acid	1.49	Hydrogen	45.02
Illuminants	3.98	Methane	37.33
Oxygen	0.28	Nitrogen	4.44

The size of coke obtained from the retorts was:

	By weight, per cent.	By measure, per cent.	Pounds per bushel
Large coke	90.96	93.70	29.86
Fine coke	3.74	3.06	37.64
Dust	5.30	3.24	50.26

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[1] London Jnl. of Gas Ltg., 1913, page 720.

Coke used to heat retorts, 13.6 per cent. of the weight of coal carbonized.

The producers are provided with rocking grates, and the fires are clinkered once in 48 hours, the power required to operate the grates being 0.6 horse power per bench.

Woodall-Duckham Verticals at Lausanne, Switzerland.—During a test run at this place 824,032 lb. of coal were carbonized, the coal being a mixture of 74.37 per cent. Márlenbach, 13.9 per cent. Altenwald, and 11.73 per cent. Heinitz, 4,595,670 cu. ft. of gas at 60°F. being produced. The result of the test, with the constructor's guaranteed figures, are given below:

	Results obtained	Results guaranteed
Yield in cu. ft. per lb. of coal	5.69	5.45
Calorific power, B.t.u.	595	570
Coke used as fuel	10.55 per cent.	16 to 18 per cent.
Gas made per retort, cu. ft.	37,505	30,900

Woodall-Duckham Verticals at La Grange, Ind.—This installation in 1910 consisted of one bench of four retorts, each retort being 25 ft. long, having a cross-section of 20 by 29 in. at the bottom and 9 by 21 in. at the top. During a 10-day run, the following average results were obtained:

Coal used	20,151 lb.
Coke made	14,841 lb.
Gas made	108,637 cu. ft.
Generator fuel used	3,598 lb.
Yield of gas per pound of coal	5.39 cu. ft.
Coke yield, per cent. of coal	73.6 per cent.
Generator fuel per 100 lb. of coal	17.89
Candle power, Sugg Argand D burner	14.47
Candle feet	77.91
B.t.u. gross	609

This bench was an experimental one, and it was impossible to maintain proper temperature conditions in the foul main and condensing apparatus.

Woodall-Duckham Verticals at Nine Elms, England.—During the run summarized below two tests of 3 days each were made, the results being:

	I	II
Coal carbonized, long tons	42.9	43.4
Gas made, cu. ft. (*a*)	524,000	548,29
Gas made per ton, cu. ft. (*a*)	12,214	12,63
Candle power (*b*)	15.83	15.8
Calorific value, gross, B.t.u.'s (*b*)	599.4	593.8
Calorific value, net, B.t.u.'s (*b*)	535.2	526.3
Carbonic acid	1.3	2.2
Sulphuretted hydrogen	1.1	2.1

An analysis of a sample of the gas from the second test collected betwee
12.00 P.M. one day and 6.00 P.M. the next, had the following composition:

Carbonic acid	2.20
Illuminants	2.88
Oxygen	0.27
Carbonic oxide	7.41
Methane	35.05
Hydrogen	45.85
Nitrogen	6.32
Candle power	16.40
Calorific value, gross, B.t.u.'s (*c*)	607
Calorific value, gross, B.t.u.'s (*d*)	609
Calorific value, net, B.t.u.'s	543

(*a*) Measured with carbonic acid and sulphuretted hydrogen in gas.
(*b*) Gas tested with carbonic acid under London regulations.
(*c*) Boy's calorimeter.
(*d*) By calculation.

The lower yield in the first test is explained by the fact that the plant was just being started up.

The Glover-West Vertical Retort.—The usual construction of this system divides the setting into two sections of four retorts each, so arranged that only half of the retorts in each setting need be in operation at a period when the total carbonizing capacity is not required. A producer is provided for each bench, the gases from the producer ascending through flues to the combustion chamber, an independent flue being provided for each half of the setting. The retorts have an oval cross-section, being 20 ft. long, and measuring 36 in. by 22 in. at the bottom, and 30 in. by 10 in. at the top. The bottom of each retort is provided with a chamber about 3 ft. deep, built of cast iron, so arranged as to form the regenerator portion of the bench. The waste gases ascending from the combustion chamber pass around the upper portions of the retorts on the way to the waste-gas flue. Fig. 44 is a diagrammatical section through the setting and typical retort house.

The bottom of each retort is provided with a coke extractor consisting of a slowly revolving vertical worm, this worm being constructed in two halves in order that one half may be detached from the other, so that, when the remaining half is partially revolved, a space for inspection and for access to the retort is provided. The extractor drive is so arranged that its speed can be varied on each individual retort, thus providing a speed adjustment suitable to any particular coal which may be carbonized in any retort. The extractor discharges the coke from the retort into a coke-receiving chamber, the latter being provided with a self-sealing door from which the coke is drawn at regular intervals.

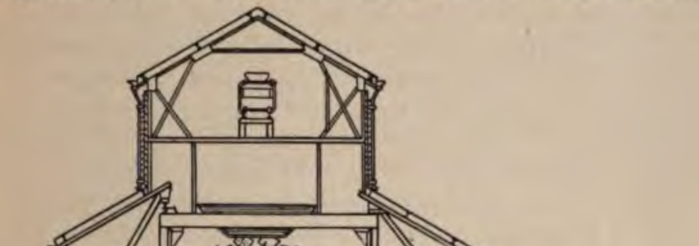

FIG. 44.—Glover-West vertical setting.

The combustion chambers are arranged in tiers, separately heated, and the products of combustion, after they have passed around the retorts, are conducted by means of vertical flues to horizontal circulating chambers which surround the upper portions of the retorts. The waste gases, as they travel up to the top circulating chambers, are caused to pass through vertical flues which adjoin the secondary air-supply flues leading to the combustion chambers, thus transmitting a portion of the waste-gas heat to the secondary air supply. The secondary air enters at the back of the setting, and circulates around the chambers located directly below the retorts, thus absorbing heat transmitted through the walls of these chambers from the coke before the latter is removed from the retorts by the extractors.

It is stated that this method of regeneration, in conjunction with the absorption of heat from the waste gases, is very effective, due to the fact that it not only cools the coke to such an extent that quenching is not required, but in that it also acts to secure fuel economy.

The gas from the retorts is conducted by means of an outlet at the base of the coal-feeding hopper through a pipe to the foul main, each outlet pipe being provided with a valve which can be closed to prevent the return of gas to the retort when the latter is shut down, or when scurfing.

The Glover-West Verticals at St. Helens.—A test made by Dr. Coleman at the St. Helens plant gave the following results:

Coal used	Wigan	Yorkshire silkstone
Yield per ton, cu. ft.	12,145	12,435
Candle power, Met. burner	15.22	16.19
Calorific power, gross, B.t.u.'s	576.2	584.9
Calorific power, net, B.t.u.'s	516.3	522.0
Coke made per ton, moist, lb.	1770.0	1714.0
Coke made per ton, dry, lb.	1557.0	1490.0
Coke used in generator moist, per cent	13.1	14.4
Coke used in generator, dry, per cent	12.2	13.4
Tar per ton, gal.	10.74	12.2
Tar, sp. gr. at 60°F.	1.075	1.084
Ammonia per ton, gal. 10 oz. liquor	28.6	23.9
Ammonia per ton, as sulphate, lb.	24.2	20.1
Sodium ferrocyanide at washer inlet per ton, lb.	5.64	5.67
Total sulphur oxide, gr. per 100 cu. ft.	30.2	27.0
Naphthalene in pur. gas per 100 cu. ft., grains	3.3	3.3
Composition of Gas.		
Carbonic acid	1.15	1.10
Unsaturated hydrocarbons	2.45	2.70
Oxygen	0.15	0.20
Carbonic oxide	7.30	7.30
Methane	33.90	32.30
Hydrogen	51.65	53.75
Nitrogen	3.25	2.40

Composition of tar	Wigan		Yorkshire silkstone	
	Volume	Weight	Volume	Weight
Light oils up to 170°C.	7.4	5.8	8.5	6.7
Middle oils, 170°C. to 270°C.	26.6	23.1	25.2	21.7
Heavy oils, 270° to 350°C.	29.5	27.8	24.1	22.5
Pitch, above 350°C.	....	42.7	....	48.5
Naphthalene	....	6.7	....	6.9
Free carbon	....	3.8	....	3.6

Glover-West Verticals at Droylsden.—Mr. J. G. Newbigging reports the following test results, page 162, made on Glover-West benches at Droylsden, in a paper read before the Institution of Gas Engineers at Glasgow, in June, 1911. Two sets of tests were made on Lancashire, on Yorkshire, and on Derbyshire coal, test A having been for the purpose of ascertaining the quantity of 17 candle-power illuminating gas which could be obtained under ordinary working conditions, while test B was made to ascertain what volume of a 15 candle-power gas could be made.

United Gas Improvement Company Verticals at Fall River.—This installation consists of seven benches, each containing nine retorts, arranged in three rows; each retort has a length of 18 ft. and a cross-section of 18 in. by 30 in. at the bottom and 12 in. by 22 in. at the top. The retorts and their heating flues are located on a steel frame 11 ft. above the lower floor level, the recuperator sections being located in the rear. The producers are also located in the rear of each setting, and they extend from the ground level to a height of approximately 13 ft., hot coke being discharged into the top by means of buggies. Coal is charged into the retorts from an overhead bunker by means of an electrically operated larry, the latter being provided with three coal hoppers and three breeze pockets. The lower doors of the retorts are of cast steel, hydraulically operated from an elevated platform, the coke from the retorts being discharged into a specially constructed car made of cast-iron plates on a steel frame, the coke from the car being deposited on a coke conveyor on a lower level.

Each setting is provided with an independent hydraulic main, each row of retorts communicating with same, the gas from the mains being led to a common header.

In heating up the retorts, the gas from the producers passes

TABLE XLII.—DROYLSDEN RESULTS

	Lancashire coal		Yorkshire coal		Derbyshire coal	
	Test A	Test B	Test A	Test B	Test A	Test B
Moisture in coal, per cent	1.62		1.34		1.50	
Sulphur in coal, per cent	1.53		1.64		3.02	
Ash in coal, per cent	3.68		7.90		8.70	
Volatile matter, per cent	33.56		30.00		31.00	
Coal carbonized, tons	129.866	140.5	150.38	93.67	130.231	78.98
Gas produced at 60°F. and 30 in., cu. ft.	1,541,249	1,831,020	1,813,420	1,333,512	1,508,691	1,126,145
Gas made per ton, cu. ft.	11,868	13,033	12,059	14,236	11,584	14,258
Candle power, Metropolitan, No. 2	17.23	15.43	17.01	15.35	17.12	15.38
Calorific value, gross, B.t.u.'s	586.6	551.8	578.8	548.2	582.4	544.0
Calorific value, net, B.t.u.'s	524.0	494.1	516.9	491.0	522.8	489.1
Sperm per ton, lb.	701.0	690.0	703.0	749.0	680.0	751.0
Candles per ton	40,897	40,220	41,025	43,704	39,663	43,857
B.t.u.'s per ton	6,961,769	7,191,609	6,979,749	7,804,175	6,746,521	7,756,352
Fuel used, per cent	9.49	9.37	10.72	12.71	9.48	10.39
Coke per ton, dry, lb.	1546.0	1534.0	1501.0	1528.0	1521.0	1467.0
Tar per ton, gal.	11.52	9.41	12.83	11.19	12.1	11.94
NH_3 liquor per ton, 10 oz. gal.	36.5	35.9	22.75	30.14	37.81	40.1
Sulphate per ton, lb.	30.7	30.2	19.1	25.3	31.8	33.72
Sodium ferrocyanide per ton, lb.	....	2.27	....	2.95	....	2.84
Sulphur oxide in purified gas, grains per 100	31.05	32.46	33.78	38.1	23.41	27.58
Naphthalene in purified gas, grains per 100	....	2.77	....	1.98	....	2.35
Analysis of Gas: Carbonic acid, per cent	2.10	2.53	0.90	1.00	1.80	2.00
Oxygen, per cent	Nil.	0.27	Nil.	0.30	Nil.	0.30
Unsaturated hydrocarbons, per cent	3.20	3.43	3.60	3.10	3.45	3.10
Carbon monoxide, per cent	9.10	10.70	6.20	6.70	9.30	10.20
Methane, per cent	31.49	26.94	30.40	28.44	31.54	27.54
Hydrogen, per cent	51.18	52.27	57.11	56.06	52.06	51.30
Nitrogen, per cent	2.93	3.86	1.79	4.40	1.85	5.56

first to a horizontal combustion chamber, located approximately on a level with the bottom section of the retort, the secondary air being admitted at this point, in consequence of which most of the combustion takes place at this point and before the gases come into direct contact with the retorts. The gases from this combustion chamber pass around the lowest section of the retorts, being directed by suitable baffles through a vertical winding path to the top, thence passing downward through the recuperators and out to the stack. A branch pipe, located between the valve at the top of the stack and the benches, leads to a turbine driven blower by means of which carbonic acid and other products of combustion may be drawn in from the stack, and thus be forced under the producer grates, while a second blower, located on the same turbine shaft, forces the secondary air to the recuperators; a by-pass is shunted between this latter line and the outlet of the carbonic acid blower for the purpose of mixing primary air with the products of combustion. This whole arrangement is governed by valves so that the mixture of waste gas and air can be regulated, and thus control the percentage of carbonic acid in the gases admitted under the grates. It is claimed that by varying this percentage the temperature of the entire setting can be raised or lowered with remarkable accuracy. Controlling valves are located in the blast pipes which conduct the "primary mixture" to the grates and the secondary air to the recuperators, for the purpose of controlling the temperatures of the individual settings; the volume of air and gas is measured by means of VENTURI meters. This arrangement obviates the necessity of inside dampers, and provides close regulation. The temperature in the hottest portion of the producer should be about 2450°F. (1343°C.), and in a test run of 30 days the minimum temperature, measured with a WANNER pyrometer, was 2318°F. (1270°C.), the maximum being 2570°F. (1410°C.), with an average of 2443°F. (1340°C.). The producers are not supplied with water pans, and no steam is admitted under the grates.

PROXIMATE ANALYSES OF COAL USED

	As received, per cent.	Dry basis, per cent.
Moisture	2.15	.
Volatile	33.80	34.54
Fixed carbon	57.50	58.77
Ash	6.55	6.69

The sulphur being 1.14 and 1.16 respectively.

RESULT OF TEST

Coal as charged, pounds	6,113,728.0
Per cent. H_2O in coal	2.20
Dry coal carbonized, in pounds	5,979,495.0
Ratio of total coke to coal, as charged, per cent.	70.6
Ratio of total dry coke to dry coal, per cent.	71.6
Bench fuel as charged, coke, pounds	848,522.0
Ratio of bench fuel to coal, as charged, per cent.	13.88
Ratio of bench fuel to coal, dry, per cent.	14.19
Gas (corrected for temperature, barometer, and holder pressure) per pound of coal, as charged, cu. ft.	5.314
Ditto, dry, cu. ft.	5.434
Candle power on Sugg D burner	16.71
Total gas made (corrected for temperature, barometer, and holder pressure) cu. ft.	32,491,200.0
Candle feet per pound of coal, as charged	88.80
Candle feet per pound of coal, dry	90.80
NH_3 produced per net dry ton, pounds	7.30
Tar (including 2 per cent. H_2O) produced per net dry ton, gallons	14.70
Average specific gravity of gas	0.4355

The tar produced was thin and easily handled, and it contained only traces of naphthalene. The analyses of this tar gave:

Specific gravity	1.127
Moisture (by volume), per cent	3.4
Free carbon, per cent	3.74
B.t.u.	15,944.0

TAR DISTILLATION TEST

Up to 170°C. (338°F.), per cent	8.00
170° to 225°C. (437°F.), per cent	12.30
225° to 270°C. (518°F.), per cent	10.30
270° to 360°C. (680°F.), per cent	23.00
Residue, per cent	48.50

U. G. I. Verticals at Hartford.—This installation consists of seven benches each of nine retorts, three retorts being placed in a row. The producers are located in the rear of the bench, Fig. 45, and are charged with hot coke from a hand car, this setting also making use of the CO_2 return system.

A test run on these benches gave the following results:

PROXIMATE ANALYSES OF COAL USED

Moisture, per cent	1.25
Volatile, per cent	36.60
Fixed carbon, per cent	55.88
Ash, per cent	6.27

The sulphur amounted to 0.876 per cent.

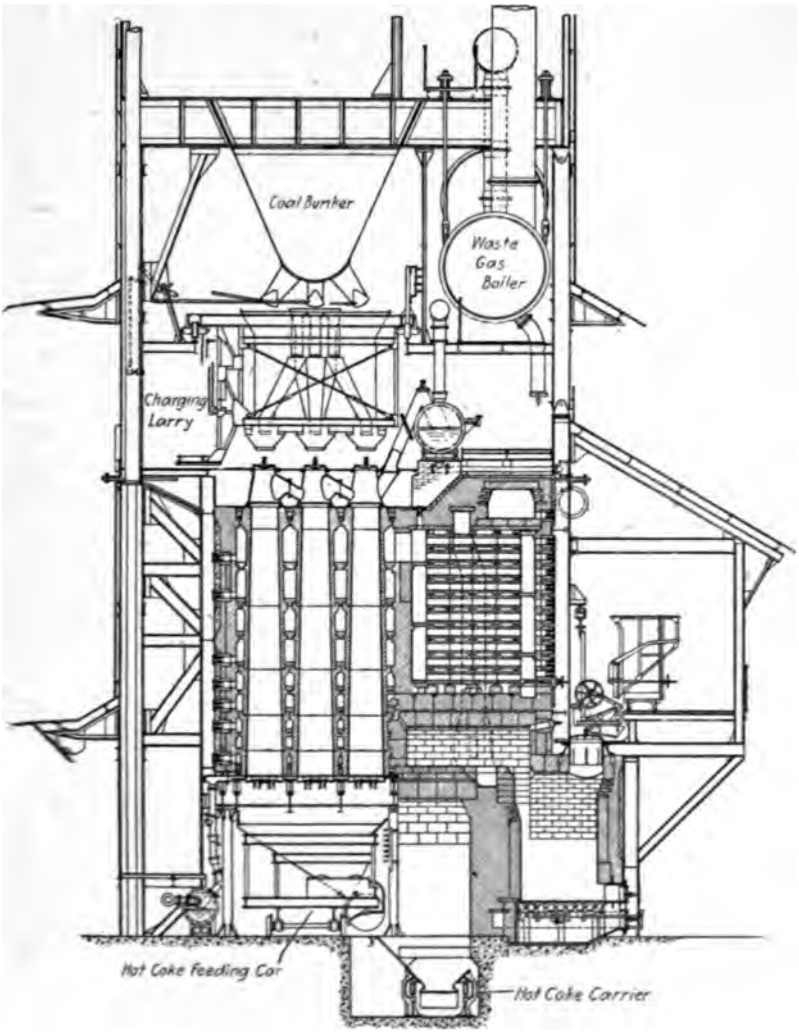

FIG. 45.—United Gas Improvement Co.'s verticals at Hartford.

RESULT OF TEST

Average gas made, 24 hours, 60°F., 30-in. bar, cu. ft.	1,132,000.0
Average yield per pound, dry basis, cu. ft.	5.37
Average length of charge, hours	11.50
Average total coke made per ton, dry basis, lb.	1,357.0
Average coke per ton, bench fuel, dry basis, lb.	263.0
Average candle power, Argand Sugg. D. burner	17.98
Average candle feet per pound dry coal	96.63
Average B.t.u. per cubic foot	635.0
Average ammonia made per ton (tested at foul main) (dry basis), lb.	7.08
Average tar made per ton of coal (dry basis) gal.	14.39

These benches were supplied with a waste-heat boiler, the average horse power developed thereby being 95. The results obtained were under the conditions of a guarantee test, and therefore do not allow for some of the operating conditions, a very important one of which is scurfing.

Retort Material.—The superiority of silica retorts and silica settings is no longer a question of doubt, and the advisability of this construction is readily conceded by all users. In referring to silica retorts care must be taken to differentiate between this material and siliceous material, the latter containing only about 85 per cent. of silica. Lime silica products, properly made and of good quality, do not always fuse, but on the contrary they are subject to considerable expansion, which causes a cracking or a flaking off of the material when subjected to a sudden change in temperature.

It is well known that lime combines quite easily with fine particles of silica at about 1100°C. (2012°F.), and, if the silica is pure, thus forming silicate of lime; but, as a rule, the silica is in combination with the oxides of alumina, iron, lime, magnesia, potash, and soda, and the greater number of these oxides present the more complex and fusible does this silicate become. It is impossible to tell at present just what chemical changes occur when such material is subjected to high temperatures, but it is a well-known fact that if the heat is intensive enough and long continued, vitrification is induced and the destruction of the material follows. Under these conditions it can be readily understood that lime is by no means the most desirable addition to make to silica aggregates, particularly if the latter be impure. Lime silica brick is, however, generally safe from fusion in the settings, but they expand more or less on further heating *in situ;* therefore, to avoid cracking or flaking under sudden changes of temperature they should be heated and cooled very slowly.

The British Refractory Materials Committee, 1913, reports that fourteen samples of retort material were obtained from ten representative manufacturers and tested for the specified requirements, these requirements being: first, a refractory test at 2960°F. (1630°C.); second, contraction or expansion 1.25 per cent. maximum; third, porosity, 18 per cent. minimum. Two of these samples failed in the test, and the committee recommends that the purchaser should always have these tests made if he desires to obtain material of uniform quality. This committee also reports that many of the bricks furnished are too friable and lacking in physical strength, and that the surfaces of the retorts become covered with fine cracks. Before the Committee prepared the test specifications given above, they obtained expert advice which indicated that much of the material was too dense and close grained, and that therefore a test for porosity would have to be inserted in the specification. It was also decided to specify that the fine dust should be excluded from the grog, and accordingly a minimum porosity of 18 per cent. was adopted, and all dust which will pass through a 16-mesh sieve is to be removed from the grog. As regards friability, it is stated that retorts are in operation which have had a life of 200 days, and which are exceedingly soft and porous, this condition being explained by the fact that the manufacturers of these retorts possessed a very intimate knowledge of their clays, and the committee therefore recommends that such manufacturers who have had to alter their mixture recently should for the present aim at obtaining a porosity of 23 to 25 per cent.

The surface cracks in the retorts are caused by the various degrees of contraction which occur in the matrix and the grog during the firing period in the kilns, but it should be possible to so produce this material that the cracks will not occur, and the porosity be secured without making the material friable.

The 1914 Committee of Progress in Carbonization Methods, American Gas Institute, states that silica retorts, with silica setting blocks for their support, are essential for modern and economical operating conditions, due to the fact that they can withstand high temperatures for several years, that they do not shrink or change their form, and that their use leads to the production of coal gas at less cost than if the coal were carbonized in clay retorts.

The silica retorts and settings have the following advantages over those of clay:

First.—Greater Durability.—This is due to the fact that the silica materials are not affected by high heats; consequently, the arches are not refilled so frequently and fewer reserve benches are required.

Second.—Less Bench Fuel.—Owing to the fact that silica has 25 per cent. greater heat conductivity than clay, higher heats can be utilized, and there is a smaller carbon deposit in the retorts.

Third.—Greater Capacity per Retort.—The higher permissible heats allow for heavier charges and a shorter period of carbonization; consequently a less number of benches are required to produce the required quantity of gas.

Fourth.—Increased Strength.—Silica maintains a uniform strength at working temperatures; consequently less benches are down for repairs at any one time.

Fifth.—Retort Alignment.—Silica retorts do not sag or bulge, because the silica materials do not shrink nor soften under high temperatures.

Sixth.—Less Scurfing Required.—Carbon does not adhere so readily to silica material, and consequently such retorts are less frequently out of commission.

A. J. REBUS[1] states that he has seen retorts built of small tongue-and-groove segments stand up under full working conditions for a period of 5 years, and he deems this length of life due chiefly to the fact that retorts of small segmental form, being flexible units, will give and take with the expansion and contraction caused by the difference in the temperatures in the settings. The impossibility of keeping the heats uniform from end to end of the bench will produce unequal contraction and expansion and, when cracks occur, complaint is usually made of the poor quality of material. In retorts built in large segments, the usual complaint refers to distortions, thus making the charge difficult of removal after carbonization. The retort built of small segments is extremely flexible, but, due to its tongued-and-grooved construction, leakage is rarely found, even after decarbonizing; if care is taken to see that the segments are hard-burned and true to shape when

[1] Am. Gas Lt. Jnl., 1912, page 233.

they are installed, no trouble will be experienced from carbon getting into the retort joints, therefore decarbonization can be as easily effected as if it were a whole or one-piece retort, and no leakage will result. Segmental retorts require good and flexible supports, and nothing is better for this purpose than the brick walls of the setting, but these walls should be of the same quality material as the retorts.

Style of Retort.—R. NUBLING[1] examined into the working conditions at Stuttgart in order to determine the character of retort best suited for the carbonization of coal. The plant consisted of thirty-two benches, each bench being provided with nine horizontal through retorts 20 ft. long and having a cross-section of 16 in. by 24 in. The heating of these settings was very uniform, the temperature in the combustion flue being maintained at 2426° to 4462°F. (1330° to 2463°C.); 1840°F. (1005°C.) in the air inlet flue; 1922° to 2066°F. (1050° to 1130°C.) in the retorts; and 1994°F. (1090°C.) in the waste gases, this latter temperature falling to 842°F. (450°C.). The CO_2 in the chimney gases amounted to from 18 to 19 per cent., and the charges varied from 1280 to 1370 lb. The coal used was 30 per cent. Saar, 40 per cent. English, 20 per cent. Ruhr, and 10 per cent. 2-year-old stock, the average daily gas yield per bench of 9's being 161,674 cu. ft.

The yield of gas per 2000 lb. of coal was 10,400 cu. ft., at 29 in. and 60.8°F., while the net heating value of the gas containing about 11 per cent. of water gas, at 30 in. and 32°F., amounted to 580 B.t.u.'s per cu. ft., and 615 B.t.u.'s for the pure coal gas.

The ground area occupied by the retort house was 29,450 sq. ft., the actual ground area of the benches proper being 8,760 sq. ft.

The cost of this plant was $157,700, or $43,800 for the retort house, $94,900 for the benches and appurtenances, and $19,000 for conveyors and appliances within the retort house. NUBLING compares this cost with verticals, ten retorts each, at $316,600; verticals, eighteen retorts each, at $250,000; Munich chambers, according to depth, at from $266,600 to $279,800. After computing these costs to the life of the different parts as a whole, he states that per 1000 cu. ft. of gas the costs pertaining to the retort house structure amount to, with vertical 18's, 4 cts., vertical 10's, 5.03 cts., chamber ovens, 4.03 cts., and horizontals, 3.19 cts. He computes the labor costs, on the basis of Stuttgart, per 1000 cu. ft. of gas, from 0.458 ct. to 0.506 ct. for the verticals, 0.458 ct. for chamber ovens, and 1.382 cts. for horizontals with straight coal gas, and 1.126 cts. with the addition of the water gas. Nubling also states that the cost of firing the benches amounts, per 1000 cu. ft. of gas, to from 6.3 cts. to 6.753 cts. for verticals, 6.52 cts. for chamber ovens, and 5.625 cts. for horizontals.

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[1] Jnl. für Gasbeleuchtung, 1912.

Nubling calls attention to the fact that the value of the by-products in general, as well as the calorific value of the gas, is about the same in all three systems, so that if there is any real economical difference, this must be sought for in the retort house construction, wages, and cost of firing; these figures total to the following in cents per 1000 cu. ft. of gas: verticals from 10.74 to 12.28; chamber ovens, 11.02; horizontals, 10.20. If steam is used in the verticals these figures will be 11.08 to 12.75, but if on the other hand, the gas made in the horizontals be reduced by sufficient water gas to bring it to the vertical retort quality, the figure for horizontal retorts and water-gas plant would be 9.76 cts.

From this it would seem that the three systems give practically the same economic results, and in consequence local reasons alone will usually determine the class of retort to be selected, but it must be remembered that the vertical retort appears to solve some of the problems of carbonization which have heretofore been accomplished in rather a crude manner, in that this latest development offers a carbonizing vessel in which the gas as evolved is not destructively overheated, in consequence of which it retains its hydrocarbons, while in the horizontal retort these same hydrocarbons are partially decomposed to form lampblack and naphthalene. Again, verticals of the Woodall-Duckham type, which are operated continuously, give a solution to the smoke problem, the smoke and soot from the usual coal-gas plant being rather a nuisance.

Charging Machines.—A fixed charge should be adopted in all retort-house practice, this charge being made to conform to the size of the retorts and to the heats maintained in the setting. This size of charge should be strictly adhered to in order to secure regular operation, the heat in the setting being maintained by proper attention to the furnace and the flues. In order to maintain this charge all coal should be weighed on scales whose accuracy is known. If the coal is delivered to the setting in hand cars, the weight of coal carried on the car should be painted on the side, as this will greatly assist in securing the desired results. The stokers should be taught that an even charge is essential to good results, and that the coal should be kept back of the iron mouthpiece.

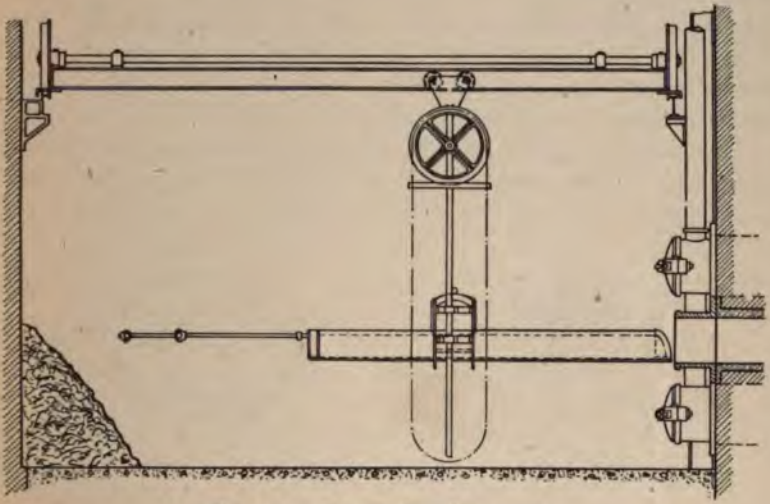


FIG. 46.—Retort charging scoop.

The charging of retorts in small and medium size works is usually accomplished with the aid of a hand scoop, which is pushed into the retort and emptied by reversing its position. This ordinary scoop is, however, subject to deformation, as the turning of the scoop usually leads to a bending of its material,

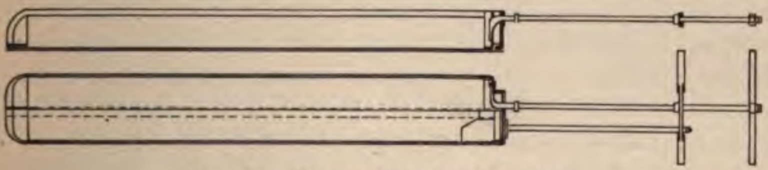


FIG. 47.—Two-section charging scoop.

this change in shape being hastened when the retort suffers a change in cross-section due to the deposit of carbon.

A very effective scoop, supported in a suspended carriage, is shown in Fig. 46, and its construction is such that it can readily lay a charge of 530 lb. of coal in a 9-ft. stop-end retort, which charge permits of filling the retort in a shorter time than usual, thus reducing the loss of gas due to open retorts. Another important object of charging is the necessity of so placing the coal in the retort that the particles are evenly distributed over the bottom and against the sides, this not being possible with the ordinary scoop. The two-section scoop, shown in detail in Fig. 47, requires the support of a basket, or hanger, while being filled, the two sections being loosely joined and liable to collapse unless supported and thus held together. The filled scoop is inserted into the empty retort as shown in Fig. 48, and as soon as the scoop is removed from its support, both halves of the scoop find support upon the bottom and against the sides of the retort, maintaining this position during progress into the retort until the position shown in Fig. 49 is assumed. When this point is reached, a lever on the handle of the scoop is turned, causing one-half of the scoop to turn to the left, and the other half to the right, Fig. 50, in such manner that both halves of the scoop will finally be in a position above each other, and over the coal, as shown in Fig. 51, occupying but little space in this position. The scoop is now withdrawn from the retort, and the halves are then reversed in direction, thus preparing it for another charge.

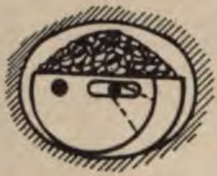


FIG. 48.

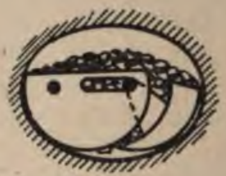


FIG. 49.

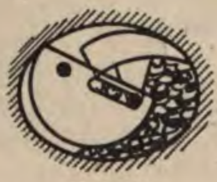


FIG. 50.

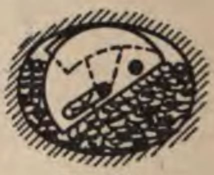
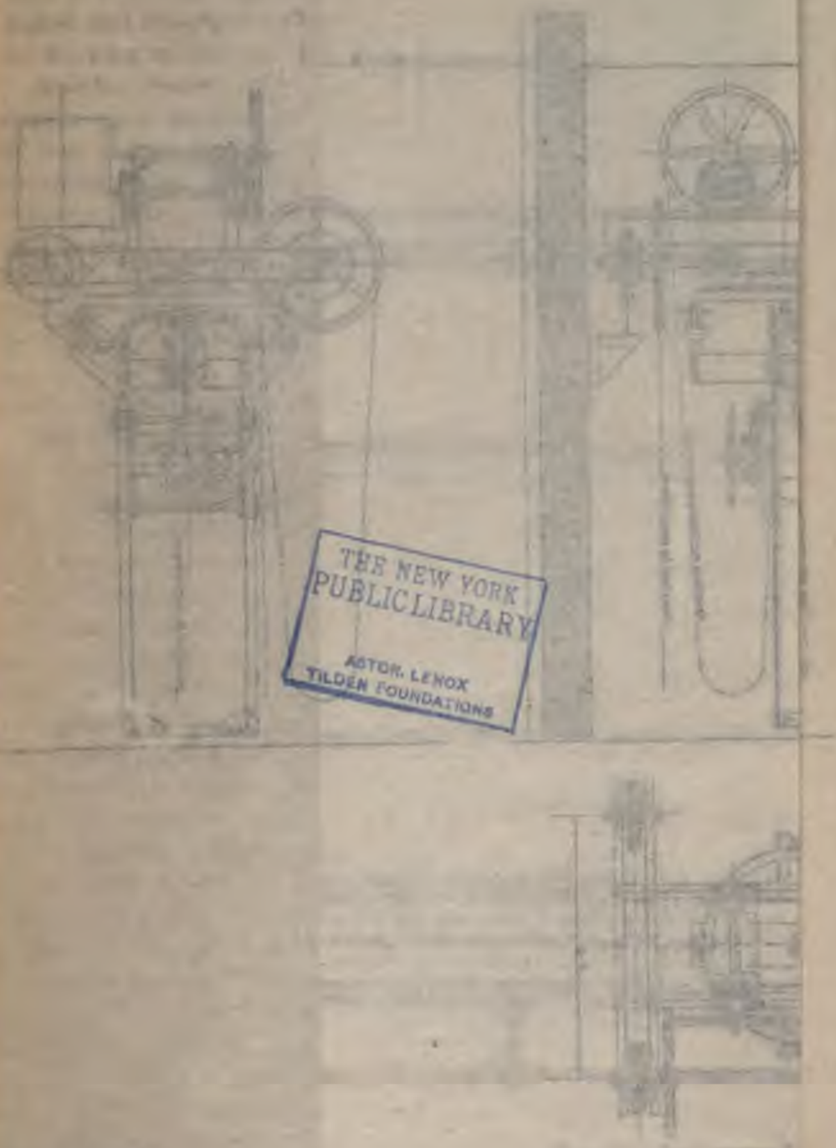


FIG. 51.

In place of the suspended crane, shown in Fig. 46, a floor carriage, Fig. 52, may be used; this carriage is so constructed that its center of gravity always passes through the wheel base, the frame being supplied with two travelling wheels, a castor being provided at front and back to prevent tipping over. Two baskets



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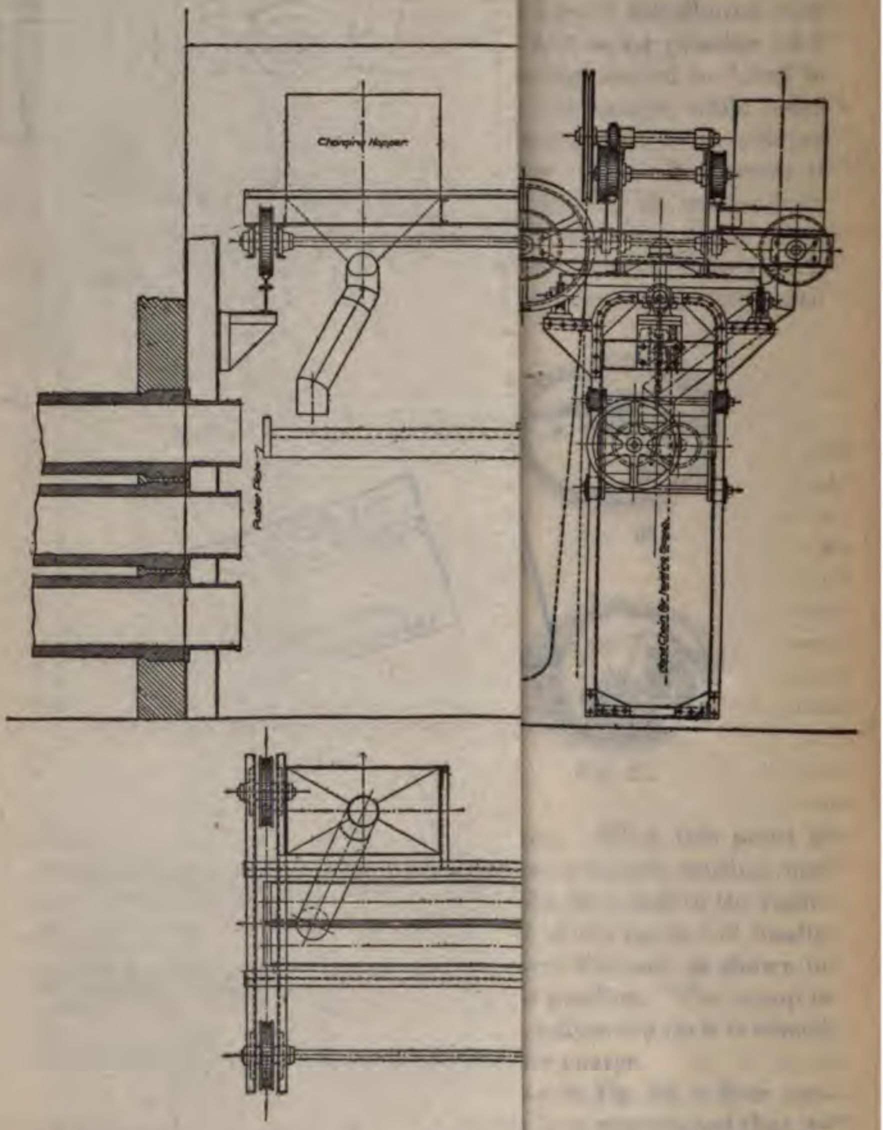

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are supplied, the stationary one at the bottom being for the support of the scoop while charging the lower retorts, the central one, raised and lowered by means of a rack and pinion, being for use in charging middle and upper retorts.

Another charger, of a semi-manual type, is shown in Fig. 53; this scoop is constructed as described above, but it is supported at one end in the operating frame suspended from an overhead traveling crane, the latter running on tracks located in front of the benches. The operating frame is attached to the crane by means of a turn-table, the scoop being raised and lowered through the medium of a hand hoist, the weight of the scoop being balanced by means of a counter-weight. The crane is moved in front of the benches by means of gearing operated by a chain, the entire machine being easily operated by one man.

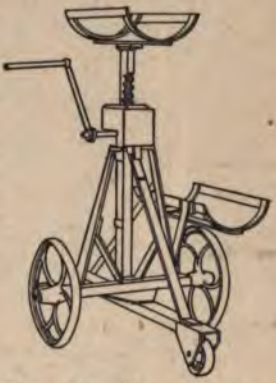


FIG. 52.—Floor carriage for scoop.

The WEST machine, as constructed in the United States, and which is a combined charger and discharger, is provided with an automatic weighing mechanism which permits of delivering a predetermined weight of coal to the retort, this mechanism being beyond the control of the operator, the volume of coal delivered from the overhead storage bins to the charging machine and from thence to the retorts, being thereby automatically weighed and recorded. When receiving coal, the operator, without leaving his station and without any other manual assistance, runs the machine under the coal bin, being careful to place it in a position which is central with the scales, after which he moves forward an electric controller which causes the hopper carried on the machine to be filled with coal, the amount being weighed and the weight being printed on a recording chart, this chart also registering the date and exact time of filling. After the hopper has been filled, the electric controller automatically returns to its initial position, and the operator then places the machine in front of the retort and proceeds to lay the charge. This automatic weighing device is sealed, no part being visible to the operator, the recording chart being removed once in every 24 hours. Great advantage is claimed for these weight records, as the manager of the plant has a daily printed

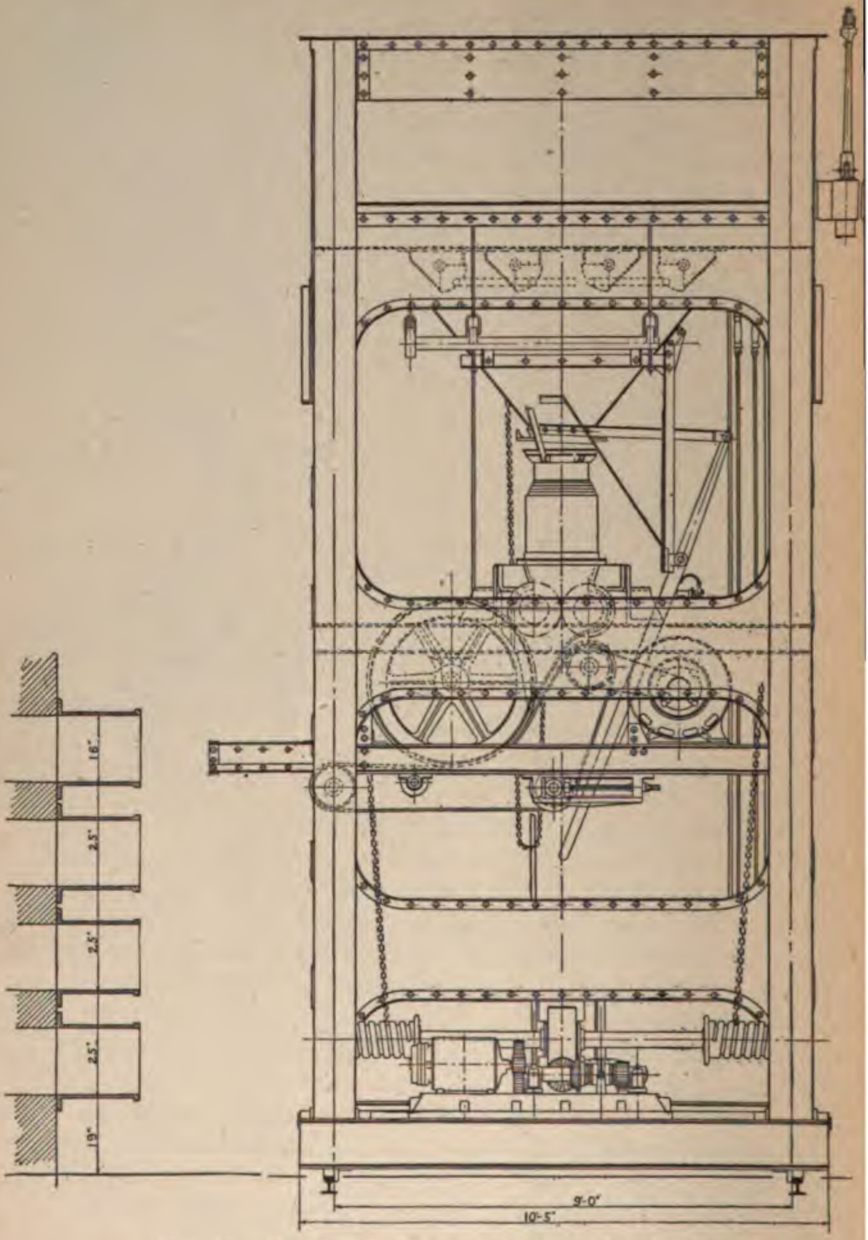


FIG. 54.—De Brouwer charging machine (side elevation).

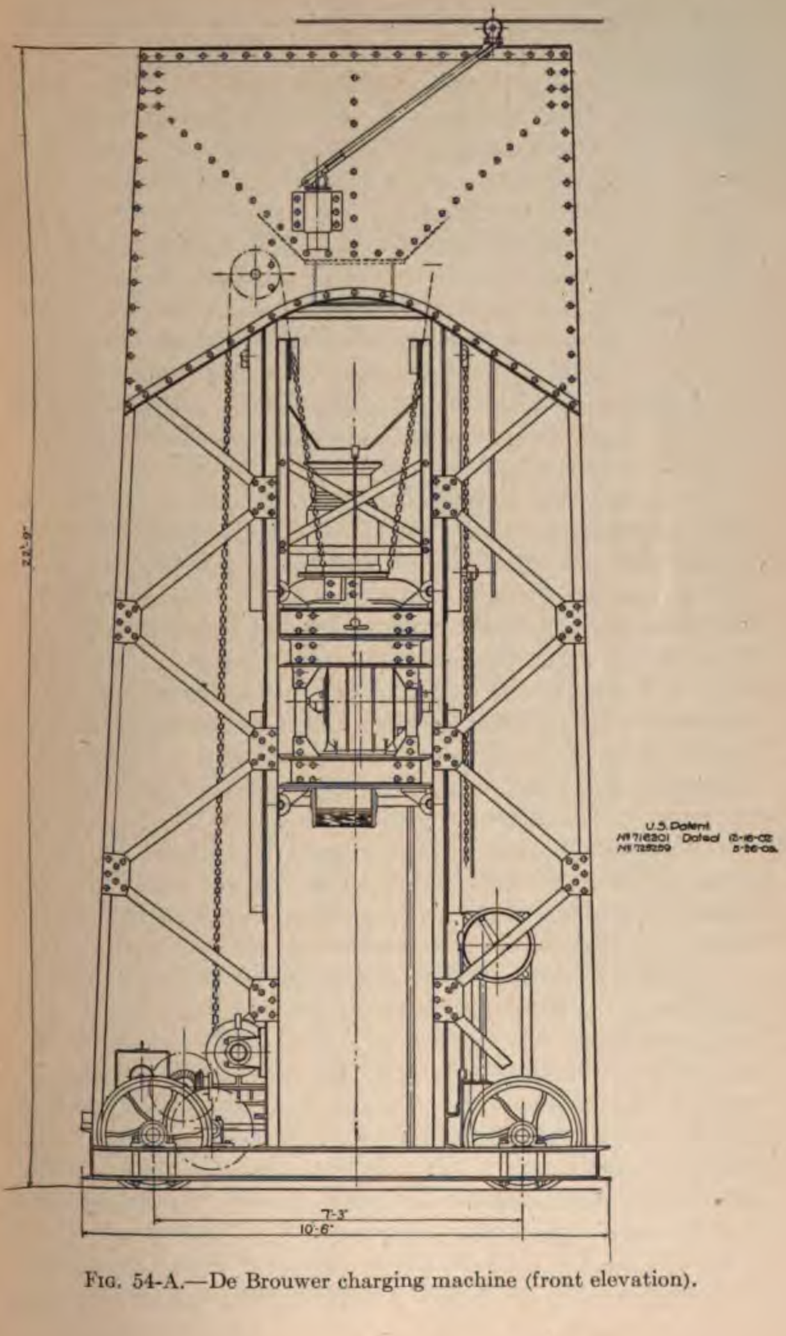


FIG. 54-A.—De Brouwer charging machine (front elevation).

record of the weight and time of each charge, this record covering all the coal carbonized.

The De Brouwer chargers and dischargers are adapted for any size plant, and they are so constructed and operated that the retort can be practically filled to the top if desired. In the adoption of these machines the retort house is usually provided with an overhead continuous coal bunker, the bottom of the bunker ending in a series of automatic valves, operated by the charger, these valves being located at convenient points in the house. In one of the usual systems adopted, the bunker is provided with a valve in front of each vertical row of retorts, and as the machine reaches this point a striker engages the valve and opens it, the charge enters the suspended weighing hopper, and as soon as the required weight is reached the machine travels forward and a second striker closes the valve.

The laying of the charge of proper cross-section with this machine is dependent upon the rate of deceleration of the speed of the charger belt, as well as upon the flow of coal upon the belt; therefore the speed of the belt must be reduced in a proper ratio from the maximum required to project the coal to the far end of the retort, to the minimum required at the near end. In the older type of machines this deceleration was accomplished by the operator, but the tendency toward longer carbonizing periods and heavier charges necessitated the mechanical control of this operation, thus eliminating the human element entirely in the laying of charges of any weight.

The charger is suspended in a rugged steel frame, Fig. 54, this frame carrying an overhead hopper having a capacity of at least 5 tons, a weighing hopper, which weighs the desired amount of coal being suspended from scales beams beneath the storage hopper. A telescopic chute leads from the weighing hopper to a pair of feeder rolls, the latter delivering a constant stream of coal to the endless charging belt, this belt passing around four pulleys. The main pulley, about 36 in. in diameter, is known as an "idler," its periphery being grooved in a manner to deliver the coal from the feeder rolls to the charging belt. The frame carrying the charger proper is caused to travel in front of the benches by means of an electric motor, this motor also being used to furnish power to raise and lower the charger to the retort to be filled.

The feeder rolls, which receive the coal from the telescopic chute, have a smooth surface which, together with the regulated speed, cause the coal to be delivered to the charging belt at the proper rate of flow, thus relieving the operator of any attention to this important feature of proper charging, as he cannot possibly feed the coal too fast, or too slow, this flow being absolutely automatic. The deceleration of the charging belt speed is also automatic, this being accomplished by means of geared mechanism shunted between the motor field rheostat and the charger proper. An important feature of this decelerating mechanism is a gear box containing an assortment of velocity ratios, due to which the time required for deceleration may be greatly varied, this mechanism being further governed by a gear with interrupted teeth, through the medium of which the rheostat arm is brought to a state of rest at a predetermined point in the charge.

This combination of automatic control of the rate at which the coal is fed to the charger, combined with the rate and time of deceleration of the charging belt, enables the machine to lay charges of uniform cross-section, and also permits of heavy charges and long carbonizing periods, as well as light charges with the consequent short carbonizing periods.

In order to lay a charge in the retort, the overhead hopper being filled with coal, the charging belt is brought from a state of rest to the low running speed, after which the machine is brought in line with the retort. The operator then rotates the rheostat arm through a small angle to the point which automatically accelerates the motor, thus bringing the charging belt to a speed of sufficient velocity to project the coal to the far end of the retort. As soon as the rheostat arm has reached this position, and while the speed of the motor is being accelerated, the operator weighs out the required charge of coal, after which he moves the arm through a small arc in the same direction as before, or until the gear with the interrupted teeth is caused to mesh with its driving mate. The coal valve in the bottom of the scales hopper is now opened wide, thus permitting the predetermined weight of coal to pass down through the telescopic chute and feeder rolls to the charging belt.

The De Brouwer discharger, shown in Fig. 55, consists of a hollow steel frame, of rectangular cross-section, the ends of the frame being semicircular, and so arranged with guides as to permit the linked ram to coil itself within the frame when in the home position. The front end of the ram is provided with a steel bar of heavy cross-section, a cast iron pusher plate being bolted to the end of this bar, this plate having a shape which closely approximates that of the retort cross-section. The underside of the bar, as well as the links, is developed into a toothed rack which meshes into a driving pinion geared directly to a motor.

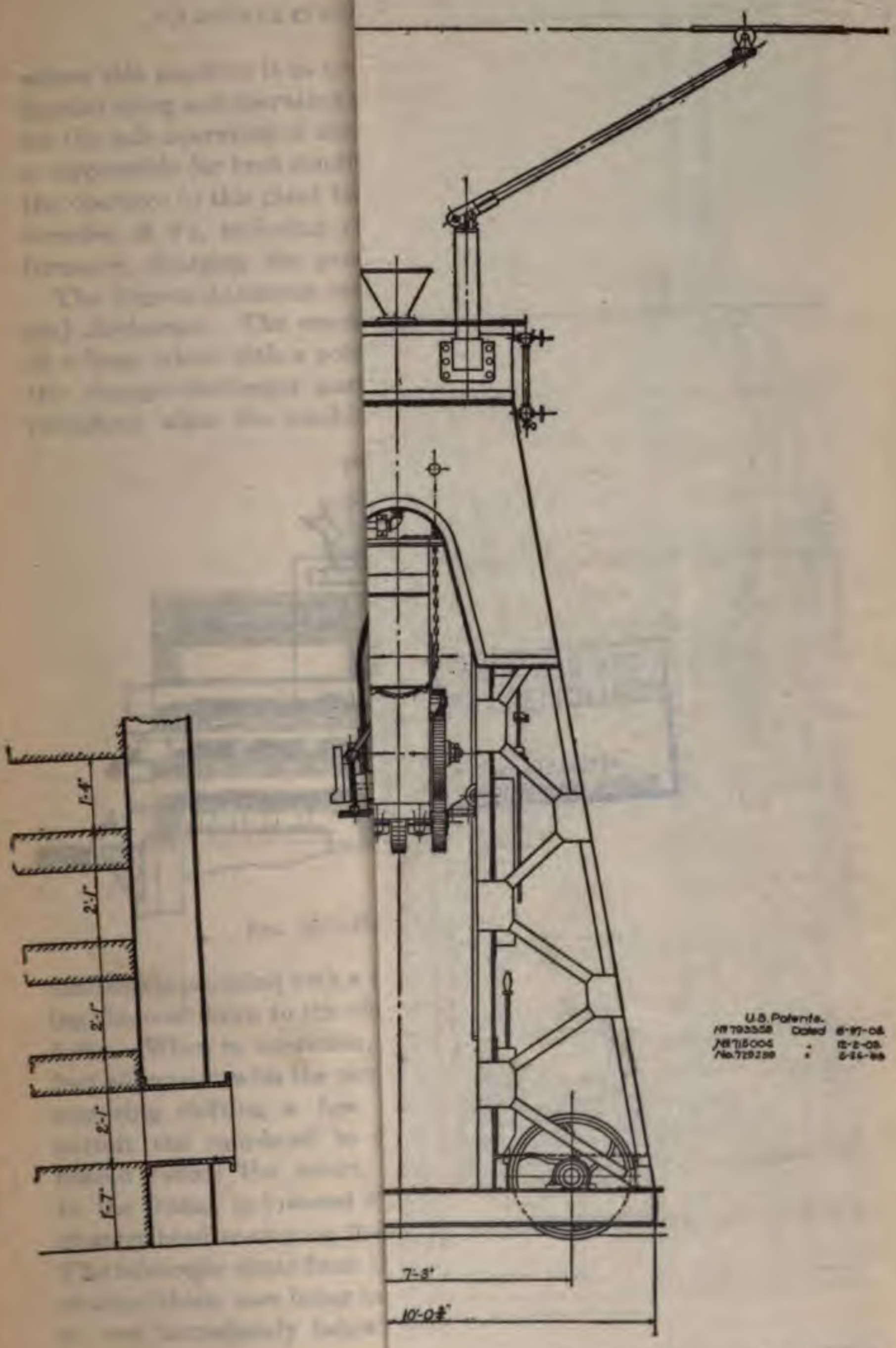
This motor is controlled by means of a master switch and a panel of automatic accelerating switches, the latter being a protection against abuse of the motor; this is necessary, because by this means the operator can abruptly throw the master switch from the forward to the reverse position, or *vice versa*, without injuring the equipment. The machine is also supplied with automatic limit switches for stopping the ram at the end of the forward stroke, returning same to the home position, and holding it there. In addition to this, mechanical safety stops are provided to prevent the ram from going forward or returning beyond a predetermined point. A water tank with flexible connections is provided and the operator is thereby enabled to spray the ram from time to time if it should become too hot.

The charging and discharging machines are each suspended from a steel floor car of very rigid construction, steel chains being used for this purpose, the raising and lowering of the machines to suit the level of the various retorts being accomplished by the same motor used for traversing the machine on the floor tracks.

The wide range in laying capacity of this machine is very apparent from the fact that they are laying from a minimum of 45 lb. to a maximum of 89 lb. per foot of retort, the time required to lay a 1200-lb. charge being about 21 seconds, while ramming the retort of the resultant coke requires about 20 seconds, the total current consumption for both charger and discharger being about 0.287 kw.-hour per ton of coal carbonized.

The De Brouwer combination charger and discharger.

In this machine the centrifugal charger, weighing hopper, scales, ram, and cooling water tank are all mounted on one frame with a 9-ft. 3-in. wheel base, the entire machine covering 120 sq. ft. of floor area, and in spite of the compactness of this machine, every part is easily accessible. The construction of the charger is similar to that described above, including coal bunker, weighing hopper, scales, etc., the discharger also being of identical construction. The principal feature in the design of this machine lies in the consolidation of control and operation of the charger, the ram, and the traverse on one platform, the arrangement being such that a high-grade operator is not necessary. In one plant



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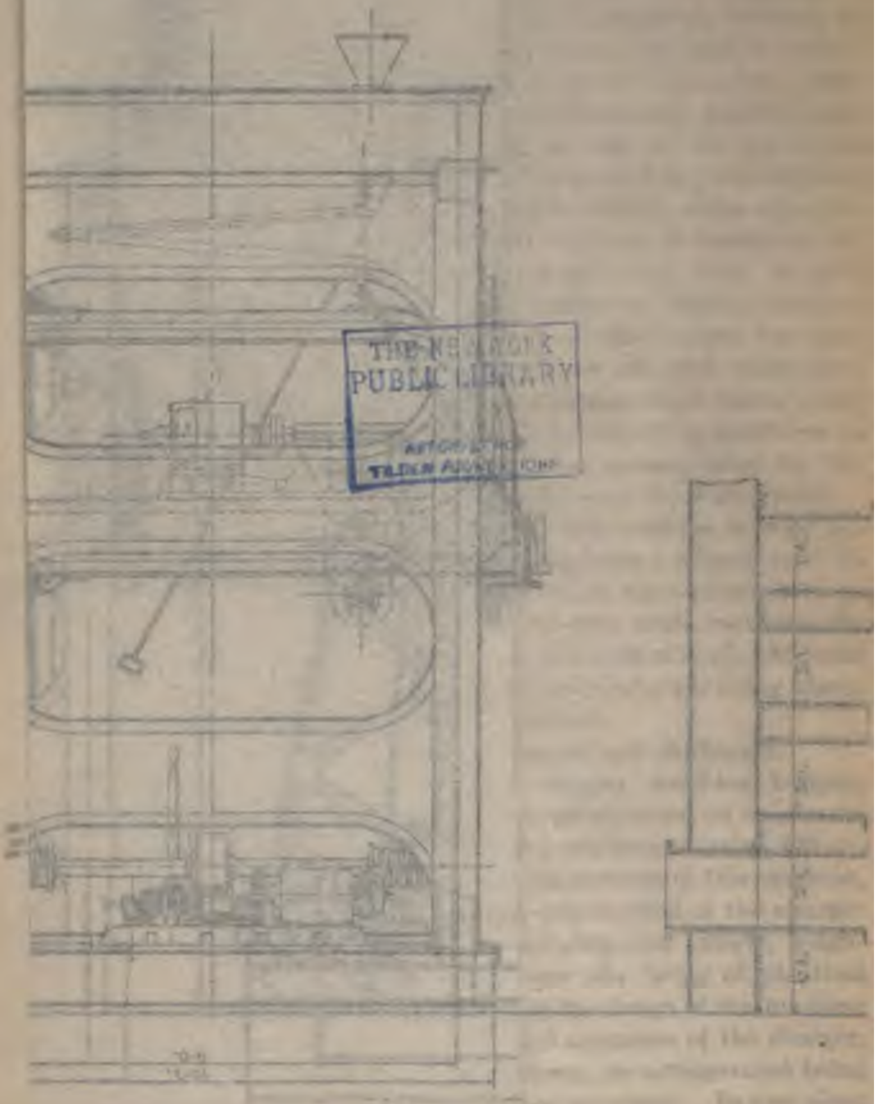

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where this machine is in use, the operator acts as shift foreman besides oiling and operating the machine, he being also responsible for the safe operation of the retort house, while the day operator is responsible for heat conditions in the benches. In addition to the operator in this plant two mouthpiece men take care of three benches of 9's, including cleaning of ascension pipes, cleaning furnaces, charging the producers, etc.

The **Fiddes-Aldridge** machine, Fig. 56, is a combined charger and discharger. The essential feature of this machine consists of a large wheel with a polygon periphery, the latter permitting the charger-discharger parts to coil about the surface of the periphery when the machine is in the "home" position. The machine is provided with a coal bunker and chute, the latter leading the coal down to the charger pans directly in front of the retorts. When in operation, this machine must be placed in perfect alignment with the center of the retort, this operation often requiring shifting a few inches to either side in order to permit the ram-head to enter. After the machine has been placed before the retort, an apron or bridge-piece, hinged to the frame, is lowered into the mouthpiece, the charger-discharger head resting on the latter until it has entered the retort. The telescopic chute from the coal hopper delivers the coal to the charger chain, care being taken to bring the sections of this chain to rest immediately below the chute, the sections being heaped

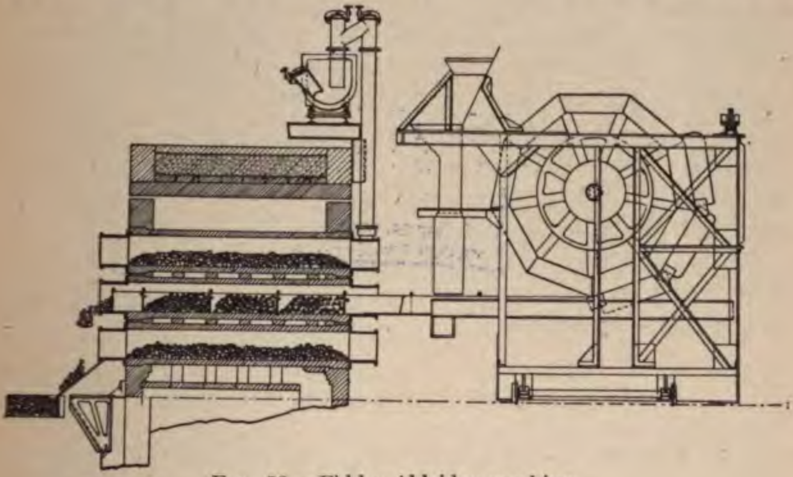

Fig. 56.—Fiddes-Aldridge machine.

charger enters the retort, the
aden chain pushes the hot coke
the charge being laid with a
end of the retort and a minimum
ference in depth being about 2 in.
construction of the charger links,
sides of the retort, nor is the coal of
he cross-section, a slight depression
The time required to lay a charge
retort, including the time required to
ort to retort, is about 120 seconds.
achine has given some excellent results
favor in this country. The operation
he installation of high-pressure hydraulic
s, and other accessories, making the in-
sive as compared with others. This ma-
rom the retorts in a manner similar to a
okes of the machine rake being required to
retort, but the operator is so located on the
inconvenienced by the hot coke.
number of other machines in use, but those
serve to depict the apparatus types in gen-
ntry.

OPERATION OF SILICA BENCHES

Baker recommends that the following method
operating coal-gas benches constructed of silica

coal-gas bench will depend largely upon the
h it has been dried out, after construction is
upon the method exercised in bringing the
proper temperature, or heat; and this statement
of benches constructed of silica, the latter
to quite some chance of damage if the heating-up
not be a slow and gradual one.
connection, **Baker** advises that, before starting the
care should be exercised to see that all flues
dirt or any obstructions left by the masons; the
should be clean, and they should contain the
of water, the body of the main being so levelled

that each dip-pipe will have the same amount of seal. The seal, when commencing operations, may be an inch or more, but after gas has been made for a few days this seal may be decreased to about ½ in. Care should also be exercised to see that the gas take-off, the tar and ammonia overflows, as well as all connections to wells, and water and steam lines to the hydraulic mains and furnaces, are all in proper operating condition.

The joints between the bottoms of the ascension pipes and the bells should be provided with a clearance of not less than 1 in. to permit of expansion, and these joints should be free and open until the heat of the bench has been raised to the charging temperature. Baker recommends that the joints be made by placing a ring of asbestos packing, or yarn, in the bottom of all mouthpiece bells, the packing material being made by mixing spent lime with fire-clay, the ring holding this material in place.

All the necessary tools required in the proper operation of the benches should be conveniently at hand, and the charging and discharging apparatus should be in proper operating condition.

In order that all heat may be conserved, and in order to prevent any sudden retort temperature changes, a common brick wall, about 4 in. thick, may be laid dry on the clay head of the retort, just inside the mouthpiece of each retort, plastering the wall up with mud, but leaving one brick in the center of the wall loose in order to provide an inspection opening, and for the purpose of giving an outlet for any accumulated gases.

Baker divides the "drying out" period into two parts, each of 5 days' duration; during this first period the *stoppers* should be removed above and below the floor line; the *dampers* and *air registers* should all be closed; the *furnace doors* and *filling doors* should be partially open; the *mouthpiece lids* should be unlatched; and the *ash pan* should be kept full of water. During this first period no steam should be admitted under the grate, and an active fire, not more than 12 in. deep, should be kept on the grate.

The fuel used should be of clean coke, hot coke being preferable, and the fire should be so replenished as to keep it burning freely, thus eliminating any possibility of an explosion of unburnt gases. Care should be taken to keep the grate free of ashes and clinkers, to maintain a uniform fire, and to avoid covering the entire fire with cold fuel.

In the second period of this process, the *stoppers* should be put in at and below the floor line; the *dampers* should be opened about 3 in.; the *air registers* should all be closed; the *furnace* ar *filling doors* should be partially open; the *mouth-piece lids* shou be unlatched; and the *ash pan* should be kept filled with wate no steam should be used under the grate, but the fuel be should be gradually deepened until it is of a depth which can b conveniently worked from the furnace door.

The "heating-up" period is also divided into two parts, eac of 7 days' duration; during the first period the *stoppers* above th floor should be put in one at a time, the bottom stopper the firs day, and the top stopper the last day; the amount of opening given the *dampers* will depend on the chimney draft; the *primary* air register should be opened about 1 in. at the start, while the *secondary* should be closed at the start, but opened to ½ in. or more each when color appears in the combustion chamber; the *furnace door* should be closed tight; the *filling* door should be partially open at the beginning of the period, but it should be closed when the secondary air registers are opened. The *mouth-piece lids* should remain unlatched; the use of water in the *ash-pan* should be discontinued, and superheated steam (if such is used) should be turned on, both the steam and the air supply being so adjusted as to maintain a slow fire. The space left open between the top of the front wall and the main arch should now be sealed tight with fire-clay putty to a depth of about 4 in. The fuel bed should be gradually increased to a depth of about 24 in., fuel being admitted from the upper or filling door.

During this period of the heating-up process the temperature increase in the setting should not exceed 100°F. (38°C.) each day until a red color begins to appear on the tiles in the combustion chamber. Careful attention must be given to the tie rods of the bench during the heating-up period, subjecting them to daily inspection, and slacking off the nuts gradually in order to prevent excessive tension in the rods. The rods should be so adjusted that there will be no appreciable tension at the start, and the total amount of slacking off should not exceed ¼ in. for each bench in half-depth settings of 6's, and ⅜ in. for each bench in full-depth settings of 6's and 8's.

During the second portion of the heating-up period, the *stoppers* should all be in place; the *dampers* should be so adjusted that there will be neither pull nor pressure at the filling door; the primary and secondary air *registers* should be adjusted to suit the increased depth of fuel; the *furnace doors* and *filling doors* should be closed tight; the *mouthpiece lids* should remain unlatched; and, if superheated steam is used under the grate, the ash pan should be dry.

During this second period the fuel in the furnace should be gradually increased in depth until the maximum is reached, and the steam and air supply should be so regulated that the temperature increase in the settings will not exceed from 150° to 200°F. (66° to 93°C.) each day. The increase in temperature should be gradual, and an excess of primary air will result in a fire which will burn too briskly.

At the end of this second period, or 24 days from the time of starting, a decided red color should appear inside the retort, after which the temporary 4-in. walls in the retorts should be removed, and a primary charge of about 250 lb. of coal should be placed in each retort, the mouthpiece lids being then closed tight. This priming charge should be kept in the retort until it is entirely burned off, and the air in the hydraulic main should be expelled before the gas valve is opened.

It is possible that the joints between the silica sections of the retorts may show a slight gas leak after a first charge of coal, but if such leaks should occur, they should not be closed with fireclay; carbon will soon close them and there should be no further trouble after the first few charges have been burned off. As the setting temperatures increase, the weight of the coal charges in the retorts should also be increased until the maximum weight has been attained.

Care should always be taken to see that the temperature increase is gradual from the initial time of starting up to the time of gas making, in order that any moisture in the masonry work will be expelled without causing damage to the setting and to permit the paper in the silica expansion joints to burn out as the tiles expand. This permits expansion to take place in each individual tile, and thus prevents the bench expanding as a whole. The drying-out period may possibly be increased to 15 days, depending upon the amount of moisture in the masonry work, and in the case of a large stack of benches the drying-out and heating-up period should be from 40 to 50 days. When an arch located between hot benches is refilled, this period may be reduced, as a partial drying out, due to radiated heat, will occur during building.

Before any adjustments for normal operation drafts are made, care should be taken to see that all bench walls, front, back, and end, are tight both below and above the floor line; any visible cracks in the walls should be pointed up, this applying particularly to walls below the floor line and around the furnace and filling doors. All recuperator flue stoppers should be sealed tight with fire-clay in order that a leakage of air into the waste-gas flues may be prevented, but the small sight-hole plugs should be left free for observation, the draft on the benches should be checked at the chimney damper, and it should be so adjusted that there is but a very slight pull on the benches.

The bench dampers should now be so adjusted that the same temperature is maintained on both sides of the bench, the draft on each bench being such that neither pull nor pressure will be occasioned at the stage floor level, and gas should blow out slightly at the stopper hole between the retorts of the second tier. If the dampers should be sufficiently closed to place a back pressure on the settings, the heat will tend to pass toward the front of the bench, while too much damper opening will cause the heat to be driven toward the back of the bench.

The primary air registers should be opened only enough to admit sufficient air to maintain a bright, live condition in the fuel bed, and upon opening the filling door each piece of coke fuel should be distinctly visible. If superheated steam is admitted under the grates through nozzles $\frac{3}{8}$ in. in diameter at about 1 lb. pressure, $\frac{1}{2}$-in. opening of each primary air slide will probably suffice. The amount of air and steam used, as well as the steam pressure employed, will depend upon the character of the fuel charged, and this must be adjusted to suit conditions.

The above adjustments should produce a temperature of not more than 1900°F. (1040°C.) below the furnace arch, and the combustion of the fuel charged will be incomplete at this point, complete combustion taking place in the combustion chamber due to the admixture of heated secondary air. The secondary air registers should be opened only sufficiently to admit the proper amount of air required to give complete combustion, about a $\frac{1}{2}$-in. opening of each secondary register probably being sufficient. A smoky or very hazy appearance in the combustion chamber will indicate an insufficient quantity of secondary air, while a very clear appearance will probably indicate an excess of secondary air.

The combustion-chamber temperature should not be much above 2400°F. (1320°C.), and this temperature should be very nearly constant throughout the chamber, a difference of 150°F. (65°C.) between front and back being the limit. The bottom waste-gas flue in a full-depth bench should exhibit no color, and it should have a temperature of less than 900°F. (480°C.), while in a half-depth bench this flue will probably show a red color and have a temperature of about 1200°F. (650°C.). When it becomes necessary to raise the heat, the primary air slides should be opened about ⅛ in. every 4 to 6 hours, and the secondary air and dampers should also be adjusted if necessary. If the heats are to be reduced, this operation should be reversed.

Baker warns against adjusting dampers and air slides at the same time, as this may cause confusion to arise when an attempt is made to determine which adjustment caused the resulting change in the bench, advising that only one adjustment be made at a time, and then waiting several hours to note results before any further adjustments are made.

When a bench is to be shut down, the last charge of coal in the retorts should be a heavy one, and it should remain in the retorts until the bench is cold. All primary air should be shut off, and the secondary air dampers should be closed gradually until the fire is out, this requiring from 4 to 6 days. The bench should now be closed tight all over, and it should be permitted to cool by radiation. The tie-rods should be watched closely, and they should be tightened if necessary, while the mouthpieces, ascension and bridge pipes should be cleaned immediately on shutting down. If the benches are supplied with one hydraulic main in common, the dip pipes should be plugged in order to prevent the escape of gas from adjoining benches.

CHAPTER VII

COMBUSTION AND THE HEATING OF SETTINGS

The term "combustion" refers to the rapid chemical action resulting in the combination of such elements as oxygen and carbon, hydrogen and sulphur, this combination being accompanied by the production of heat and light. In this combination we find that the substance which combines with the oxygen is the combustible, but the oxygen and nitrogen contained in the volatile matter of the fuel, while not being combustible, have, through custom, been given this name, and therefore the term, "combustible," in its present accepted sense, applies to that portion of the fuel which is dry and free from ash, thus including oxygen and nitrogen.

Perfect combustion is secured when the combustible is oxidized to the highest possible degree, or when carbon is completely converted into carbon dioxide (CO_2); if the carbon should be converted into carbon monoxide (CO), the combustion would be imperfect, because the CO can be burned into CO_2.

In order to produce combustion, or the union of the oxygen with the combustible, a certain degree of heat, or a kindling temperature is required, the approximate kindling temperatures for various substances being given below:

KINDLING TEMPERATURES

Lignite dust.....	300°F.	149°C.
Dried peat.....	435°F.	224°C.
Anthracite dust..	570°F.	300°C.
Anthracite.....	750°F.	400°C.
Bituminous coal.	600°F.	315°C.
Coke....	Red heat	Red heat
Sulphur...	470°F.	243°C.
Carbon monoxide	1211°F.	655°C.
Hydrogen	1030°F. to 1290°F.	555°C. to 700°C.

The chief elements to be considered in the matter of combustion are oxygen (O), carbon (C), hydrogen (H), sulphur (S), and nitrogen (N). The oxygen is taken from the air entering the furnace; it has a density of 1.10521 (air being 1); a weight of

0.088843 lb. per cubic foot at 32°F. and atmospheric pressure; and an atomic weight of 16. One pound of air contains 0.2315 lb. of oxygen, or 1 lb. of oxygen is found in 4.32 lb. of air.

Carbon is found in all fuels, and it is the most abundant of all combustibles; its atomic weight is 12.

Hydrogen is found at times in small quantities in some fuels as a free constituent, but in most cases it is found in combination with the carbon. It has a density of 0.0692 (air being 1); a weight of 0.00559 lb. per cubic foot at 32°F. and atmospheric pressure; and an atomic weight of 1.

Sulphur is found in a lesser or greater degree in nearly all coals, it being present in most cases as either sulphide of iron, or sulphate of lime, it having no heating value in the latter form. Sulphur has an atomic weight of 32.

Nitrogen has no useful effect in combustion, it being a necessary evil emanating from the air drawn into the furnace; it has a diluent effect on the required oxygen, absorbs heat and thus reduces the temperature of the products of combustion. It has a density of 0.9701 (air being 1); a weight of 0.07831 lb. per cubic foot at 32°F. and atmospheric pressure; and an atomic weight of 14. Each pound of air at atmospheric pressure contains 0.7685 lb. of nitrogen, and 1 lb. of nitrogen will be found in 1.301 lb. of air.

The combination of these various elements to produce combustion requires definite proportions of each, these proportions always being the same, the quantity of heat produced by this union also being invariable, no matter whether the combustion is completed in one or two stages, as will be seen by the following reactions:

Carbon to carbon dioxide:

Combining elements.	Molecular weights.	Lb. of elements.
$C + O_2 = CO_2$,	or $12 + 32 = 44$,	or $1 + 2.666 = 3.666$

Carbon to carbon monoxide:

$C + O = CO$,	or $12 + 16 = 28$,	or $1 + 1.333 = 2.333$

Carbon monoxide to carbon dioxide:

$CO + O = CO_2$,	or $28 + 16 = 44$,	or $1 + 0.571 = 1.571$

Table XLIII gives the combustion data for carbon, hydrogen, methane, sulphur, etc.

the amount of oxygen introduced as aqueous vapor, and consequently a reduction in the amount of air supplied to the fuel in the furnace must be considered when making combustion calculations under this latter condition.

During complete combustion the hydrogen will be converted into water vapor, but not all of the hydrogen given by the fuel analysis is available for heat production, because the oxygen given by the analysis is united with a portion of the hydrogen in the form of water, and was therefore already in combination before combustion was effected. As the combining weights of the two elements in water are 1 and 16, the weight of the combined hydrogen will be one-eighth that of the oxygen, or the hydrogen available for combustion will be $H - \frac{O}{8}$.

The heat of combustion of the fuel used can be determined with the aid of DULONG's equation, or

$$14{,}544C + 62{,}028\left(H - \frac{O}{8}\right) + 4050S \qquad (35)$$

in which C represents the carbon, H the hydrogen, O the oxygen, and S the sulphur constituents, by weight, of the fuel. If the fuel should contain carbon monoxide this equation would have to be altered by adding the term 10,150C, in which the proportionate weight of carbon which is converted into carbon monoxide is represented by C; if the weight of the carbon monoxide is known, the equation is altered by adding 4350CO.

The following example is presented in order to show the determination of a fuel's heating value, assuming that the fuel has the following analysis:

Hydrogen	4.25 per cent.
Carbon	75.31 per cent.
Oxygen	5.60 per cent.
Sulphur	3.35 per cent.
Nitrogen	1.18 per cent.
Ash	10.31 per cent.
	100.00 per cent.

Applying these figures to DULONG's equation, we have

$$14{,}544 \times 0.7531 + 62{,}028\left(0.0425 - \frac{0.0560}{8}\right) + 4{,}050 \times 0.0335,$$

equivalent to $10{,}953 + 2{,}202 + 136 = 13{,}291$ B.t.u.'s.

The amount of oxygen required to produce combustion with above fuel is given by:

$$C \times 2.666 + \left(H - \frac{O}{8}\right) H \times 8 + S \times 1$$

or

$$0.7531 \times 2.666 + \left(0.0425 - \frac{0.0560}{8}\right) 0.0425 \times 8 + 0.0335$$

equivalent to $2.008 + 0.012 + 0.0335 = 2.0535$ lb.

The amount of air required to furnish this amount of oxyg will be $2.0535 \times 4.32 = 8.871$ lb. and as the total weight of co bustible amounts to $C + \left(H - \frac{O}{8}\right) + S$, or $0.7531 + 0.0355$ 0.0335, or 0.8221 lb., the amount of air required per pound combustible will be $\frac{8.871}{0.8221} = 10.79$ lb.

If it is desired to produce perfect combustion with this quantity of air, it would be necessary to bring each particle of the contained oxygen into direct and intimate contact with the fuel, which, as stated before, is not possible in practice, and as much as 50 per cent. excess air is required at times. A low supply of air does not permit of burning the carbon to dioxide, but monoxide results, and thus reduces the available calorific value. Too much air is also harmful, because it acts as a diluent to the products of combustion and also reduces the available temperature by absorbing heat which is carried away with the waste gases.

As air is a perfectly elastic body, it will be self-evident that the density of the atmosphere will decrease in a geometric ratio with the altitude, and consequently the weight and volume of air will depend upon pressure and temperature, or

$$Pv = 53.3T \tag{37}$$

in which

P = the absolute pressure, in pounds per square foot.
v = volume occupied by 1 lb. of air in cubic feet.
T = the absolute temperature of the air, in degrees F.

In accordance with equation (37), we find that the weight of 1 cu. ft. of air will be given by

$$W = \frac{1}{v} \tag{38}$$

As an example of weight determination, we will assume the air to be under a gauge pressure of 59 lb. to the square inch with a temperature of 110°F. Under these conditions the absolute pressure will be

$P = 144\ (14.7 + 59) = 10{,}613$ lb. per square foot; the absolute temperature will be

$T = 110 + 461 = 571°$, and

$v = \dfrac{53.3 \times 571}{10{,}613} = 2.868$ cu. ft., making the weight of 1 cu. ft.

$W = \dfrac{1}{2.868} = 0.348$ lb.

This same procedure holds good for nitrogen, hydrogen, and oxygen, but instead of the constant 53.3, the constant 54.926 should be used for nitrogen; 770.322 for hydrogen; and 48.257 for oxygen.

Table XLIV gives the volume and weight of air at atmospheric pressure, and at various temperatures.

TABLE XLIV.—VOLUME AND WEIGHT OF AIR AT ATMOSPHERIC PRESSURE AND AT VARIOUS TEMPERATURES

Temperature		Volume of 1 lb. in cubic feet	Weight of 1 cu. ft. in pounds	Temperature		Volume of 1 lb. in cubic feet	Weight of 1 cu. ft. in pounds
°C.	°F.			°C.	°F.		
10	50	12.840	0.077884	110	230	17.362	0.057596
13	55	12.964	0.077133	116	240	17.612	0.056774
15	60	13.090	0.076400	121	250	17.865	0.055975
18	65	13.216	0.075667	127	260	18.116	0.055200
21	70	13.342	0.074950	132	270	18.367	0.054444
24	75	13.467	0.074260	138	280	18.621	0.053710
27	80	13.593	0.073565	143	290	18.870	0.052994
29	85	13.718	0.072894	149	300	19.121	0.052297
32	90	13.845	0.072230	160	320	19.624	0.050959
35	95	13.970	0.071580	171	340	20.126	0.049686
38	100	14.096	0.070942	182	360	20.630	0.048476
43	110	14.346	0.069698	193	380	21.131	0.047323
49	120	14.598	0.068500	204	400	21.634	0.046223
54	130	14.849	0.067342	218	425	22.262	0.044920
60	140	15.100	0.066221	232	450	22.890	0.043686
66	150	15.352	0.065140	246	475	23.518	0.042520
71	160	15.603	0.064088	260	500	24.146	0.041414
77	170	15.854	0.063072	274	525	24.775	0.040364
82	180	16.106	0.062090	288	550	25.403	0.039365
88	190	16.357	0.061134	300	575	26.031	0.038415
93	200	16.606	0.060210	315	600	26.659	0.037510
99	210	16.860	0.059313	344	650	27.913	0.035822
100	212	16.910	0.059135	370	700	29.172	0.034280
104	220	17.111	0.058442	400	750	30.428	0.032865

A small amount of steam is usually admitted underneath the grates in the producer for the purpose of, first, to minimize the formation of clinker by reducing the temperature in the clinker zone to a point which lies below the fusing temperature of the ash, in this manner securing more complete fuel combustion, as well as a better distribution of the primary air; and second, the steam is decomposed in passing through the bed of fuel, thus forming water gas, which in turn gives a higher heat at the point of combustion, the latter depending upon the percentage of H_2 produced, and in this manner reducing the volume of primary air required.

The steam used is generated and admitted into the producer in several ways, the two methods most commonly practised being the trickling of water on the grates, which then collects in the ash pans and is evaporated by the radiant heat, or by generating the steam in the boiler located in the waste-gas flues; a third method is the admission of steam generated by some external means. The first two methods are usually preferred to the latter, because the steam generated in either of these two ways contains more moisture than that produced by outside source, and is therefore of more value; besides this, an excessive amount of steam is thereby avoided. If steam from outside sources is used, proper regulating valves must be supplied, as an excessive amount of steam soon lowers the temperature in the producers, with a consequent poor quality of fuel gas when the temperature drops below 1800°F. (980°C.). This latter condition will also cause the production of a considerable amount of clinker at some distance above the grate level, owing to the raising of the zone of carbon dioxide reduction. It is impossible to give any hard-and-fast rule regarding the amount of steam to be used, as this will depend entirely upon the fusing point of the ash, as will be explained later, but no more should be used than is sufficient to maintain good furnace operation. In some works from 1 to 1.5 lb. of steam per pound of fuel is used, and again this may be as low as ½ lb., all depending on the character of the fuel and its consumption per square foot of grate in a unit of time.

In some installations the waste gases from the chimney are returned to the producer, the operating proportions being 50 per cent. gases and 50 per cent. air, this method of operation producing results in the furnace similar to the admission of steam, the intention being to secure fuel economy by utilizing the carbon content of the flue gas.

Before any heat calculations can be made, it is necessary to know the specific heat of the substances under consideration, and specific heat may be defined as the ratio between the quantity of heat required to raise the temperature of the unit weight of any substance through 1°F., as compared with the amount of heat necessary to raise the same weight of water through the same range of temperature. The specific heat of gases varies, and it is necessary to make a distinction between specific heat at constant volume and at constant pressure; the specific heats of various gases are given in Table XLV.

TABLE XLV.—SPECIFIC HEAT OF GASES

Gas	At constant pressure	At constant volume
Air (at freezing point)	0.2375	0.1685
Oxygen	0.2175	0.1551
Nitrogen	0.2438	0.1727
Hydrogen	3.4090	2.4123
Carbon dioxide (CO_2)	0.2170	0.1535
Carbon monoxide (CO)	0.2479	0.1758
Olefiant gas	0.4040	0.1730
Blast-furnace gas	0.2277	....
Chimney gas (approx.)	0.2400	....
Sulphurous acid	0.1550	....
Coke	0.2000	....
Ashes (approx.)	0.2000	....
Firebrick (approx.)	0.1000	....
Superheated steam	0.4805	0.3460

In the usual gas producer practice the specific heat at constant pressure only is considered, and as carbon dioxide and water vapor enter into these calculations to a great degree, it is necessary to know the specific heats of these substances at various temperatures; these heats are given in Table XLVI.

Bench settings are provided with furnaces or producers which are usually classified as "direct-fired" and "gaseous-fired," these two classes being further subdivided with reference to the admission of air, the subdivisions for the direct-fired producer embodying, first, those which are provided with a full supply of air under the grate; and, second, those with a partially controlled air supply. The gaseous-fired producers consist of, first, those in which both primary and secondary air are under the control of the operator; second, those producers which are supplied with secondary air, the latter being heated by contact with the hot walls of the producer; third, those in which the secondary air is heated through the medium of the waste gases; and, fourth, those in which both the primary and the secondary air supply are heated by the waste gases.

TABLE XLVI.—MEAN SPECIFIC HEATS OF CARBON DIOXIDE AND WATER VAPOR

Temperature		Carbon dioxide	Water vapor
°C.	°F.		
0	32	0.184	0.420
100	212	0.192	0.438
200	392	0.200	0.456
400	752	0.215	0.492
600	1112	0.228	0.529
800	1472	0.240	0.565
1000	1832	0.251	0.602
1200	2192	0.261	0.638
1400	2552	0.270	0.674
1600	2912	0.278	0.711
1800	3372	0.286	0.747
2000	3632	0.293	0.784
2200	3992	0.300	0.820
2400	4352	0.306	0.856
2600	4712	0.312	0.893
2800	5072	0.318	0.929

In the direct-fired bench, with a full supply of air under the grate, the entire amount of air required for combustion is admitted into the furnace beneath the grate bars, combustion being completed on the grate, no means being supplied admitting of air control. In the second classification of this class of bench the furnace is so modified that the primary air is placed under partial control, this modification not only improving the fuel results, but also leading to a better distribution of the resultant heat, due to the fact that a lesser amount of the products of combustion require heating up than is the case with the first class. Some of the producers or furnaces of the second class have also been constructed in such manner that the secondary air is brought under the control of the operator, but in both types the fuel bed remains so shallow that the furnace cannot be classed under the producer type, and a great deal of clinker is formed, the latter condition requiring frequent clinkering periods.

It is impossible to maintain even heats on benches fired in this manner, and the retort close to the furnace will always be found with a higher temperature than those further removed; to this unfavorable condition must be added that due to the admission of an excess of air while the furnace is being clinkered. After the heated products of combustion have passed through the setting they are usually discharged into the atmosphere with a temperature of about 1800°F. (970°C.); due to this fact the fuel economy which would result by the utilization of these products in preheating the primary air cannot be obtained, it being claimed[1] that 50 per cent. of the theoretical heat of the fuel is thus lost when 20 per cent. excess of air is used to produce combustion. This loss will be exceeded when the chimney gases contain more than 4.3 per cent. of oxygen and less than 16.3 per cent. of carbonic acid, and when this condition is experienced it becomes necessary to reduce the supply of air to the furnace until

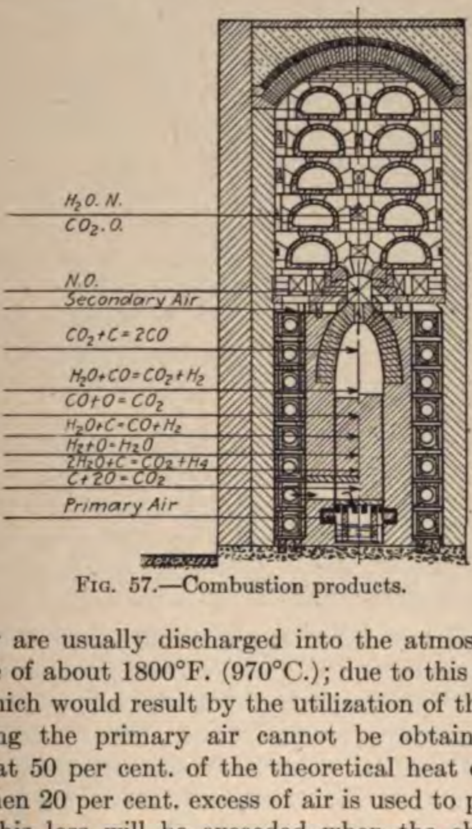

FIG. 57.—Combustion products.

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[1] Gas Manufacture, W. J. A. BUTTERFIELD, page 80.

the above percentages are realized, because the presence of mo than 4.3 per cent. of oxygen in the waste gases of a direct-fir furnace indicates a low thermal efficiency.

The modern gaseous-fired benches are due in great measure t the experiments of SIEMANS, SCHILLING, LIEGEL, and KLÖNN their work in this direction having led to present fuel econom and carbonizing results. In the gaseous-fired bench the combustible gases, carbon dioxide and hydrogen, are generated in producer so designed that a fuel depth of sufficient dimensions fo the production of a continuous supply of gases is always maintained. Fig. 57 is a section through a gaseous-fired bench illustrating the burning of the fuel, with its consequent products and the heating and admission of both primary and secondary air, the object of the arrangement being the restriction of the primary air to such an extent that only a partial combustion of the carbon takes place in the furnace, complete combustion being accomplished in the combustion chamber by the admission of secondary air, this final combustion being so regulated, by means of dampers, that it is caused to take place at any desired point in the setting, thus regulating the heats to greater advantage. This arrangement results in greatly improved furnace conditions, producing a more complete combustion of the fuel with a decreased production of clinker, greater thermal efficiency, and decreased labor.

In the earlier constructions of this type of producer, the secondary air was usually admitted at the top of the fuel bed, this air being heated by contact with the hot producer walls in some instances; but in this case the total resultant heat was about the same as that secured with a direct-fired bench, because the air received its heat from the producer at the expense of the heat in the producer gas, and therefore this type of furnace cannot be really termed a generator. The only advantages possessed by this method of construction lie in the fact that furnace conditions are somewhat improved and, due to a closer regulation, there is less possibility of supplying more air than conditions warrant.

Under the term of true recuperators we find only two classes of furnaces, those in which the primary air, and those in which both primary and secondary air are heated through the medium of the waste gases, because in both cases heat, which would otherwise be lost, is used to advantage. This construction is also termed "regenerative" at times, but not truly so, because in the regenerative system, as exemplified in the SIEMEN furnace, the air is heated by means of stored heat, the flow of both waste gas and air being periodically reversed, the incoming air being thus heated by the heat stored in the flues through the passage of the waste gases. In contradistinction to this system, we find the recuperative method to be a continuous one, because the same flue is used at all times for the same service, the other system being intermittent. In the GLOVER-WEST system of vertical retorts we find the *secondary* air circulating around the lower chambers of the recuperator, thus absorbing heat from the coke which is about to be discharged, while in the WOODALL-DUCKHAM system the *primary* air is heated in a similar manner. This method of operation has two advantages in that it cools the coke to such a degree that no, or very little, water is required for quenching, and it also reduces the fuel consumption to a considerable extent. The arrangement of recuperator flues is not a standard one, each bench designer varying them according to individual ideas, but the endeavor of all is to provide a system of recuperation which will give a maximum travel to both gas and air, flowing in opposite directions, thus making the greatest use of the waste gases.

Besides thus utilizing the waste gases, late developments have also made use of the escaping heat in the generation of steam in boilers of the water-tube type, settings of this character being operated under induced draft.

The temperature of the fire can be determined, when the heat due to the combustion of the fuel and the weight, as well as the specific heat of the products of combustion, are known (without taking into account the heat lost by radiation and conduction) by the equation

$$\text{B.t.u.} = W \times H \times T \tag{39}$$

in which

B.t.u. = the heat of combustion.
W = weight of the products of combustion.
T = temperature elevation of the products in degrees F.
H = specific heat of the products.

If we neglect the sulphur, and the small quantity of oxygen required to burn it, in the fuel previously cited, and assume that the fuel is burned with the minimum amount of air, the total weight of the products of combustion will be:

with coal in this manner. As the charger enters the retort, the ram-head on the end of the coal-laden chain pushes the hot coke out of the discharge mouthpiece, the charge being laid with a maximum depth at the discharge end of the retort and a minimum depth at the near end, this difference in depth being about 2 in. The coal in the retort, due to the construction of the charger links, does not lie close against the sides of the retort, nor is the coal of an even depth throughout the cross-section, a slight depression being noticeable in the middle. The time required to lay a charge of 65 lb. of coal per foot of retort, including the time required to move the machine from retort to retort, is about 120 seconds.

The FOULIS drawing machine has given some excellent results but has not found much favor in this country. The operation of this machine requires the installation of high-pressure hydraulic pipelines, accumulators, and other accessories, making the installation quite expensive as compared with others. This machine draws the coke from the retorts in a manner similar to a hand rake, several strokes of the machine rake being required to completely empty the retort, but the operator is so located on the machine that he is not inconvenienced by the hot coke.

There are quite a number of other machines in use, but those described above will serve to depict the apparatus types in general use in this country.

THE OPERATION OF SILICA BENCHES

MR. VERNON BAKER recommends that the following method be adopted for operating coal-gas benches constructed of silica material.

The life of any coal-gas bench will depend largely upon the manner in which it has been dried out, after construction is completed, and upon the method exercised in bringing the bench up to its proper temperature, or heat; and this statement is particularly true of benches constructed of silica, the latter being subject to quite some chance of damage if the heating-up process should not be a slow and gradual one.

In this connection, BAKER advises that, before starting the fires on the grates, care should be exercised to see that all flues are free from dirt or any obstructions left by the masons; the hydraulic mains should be clean, and they should contain the proper height of water, the body of the main being so levelled that each dip-pipe will have the same amount of seal. The seal, when commencing operations, may be an inch or more, but after gas has been made for a few days this seal may be decreased to about ½ in. Care should also be exercised to see that the gas take-off, the tar and ammonia overflows, as well as all connections to wells, and water and steam lines to the hydraulic mains and furnaces, are all in proper operating condition.

The joints between the bottoms of the ascension pipes and the bells should be provided with a clearance of not less than 1 in. to permit of expansion, and these joints should be free and open until the heat of the bench has been raised to the charging temperature. Baker recommends that the joints be made by placing a ring of asbestos packing, or yarn, in the bottom of all mouthpiece bells, the packing material being made by mixing spent lime with fire-clay, the ring holding this material in place.

All the necessary tools required in the proper operation of the benches should be conveniently at hand, and the charging and discharging apparatus should be in proper operating condition.

In order that all heat may be conserved, and in order to prevent any sudden retort temperature changes, a common brick wall, about 4 in. thick, may be laid dry on the clay head of the retort, just inside the mouthpiece of each retort, plastering the wall up with mud, but leaving one brick in the center of the wall loose in order to provide an inspection opening, and for the purpose of giving an outlet for any accumulated gases.

Baker divides the "drying out" period into two parts, each of 5 days' duration; during this first period the *stoppers* should be removed above and below the floor line; the *dampers* and *air registers* should all be closed; the *furnace doors* and *filling doors* should be partially open; the *mouthpiece lids* should be unlatched; and the *ash pan* should be kept full of water. During this first period no steam should be admitted under the grate, and an active fire, not more than 12 in. deep, should be kept on the grate.

The fuel used should be of clean coke, hot coke being preferable, and the fire should be so replenished as to keep it burning freely, thus eliminating any possibility of an explosion of unburnt gases. Care should be taken to keep the grate free of ashes and clinkers, to maintain a uniform fire, and to avoid covering the entire fire with cold fuel.

In the second period of this process, the *stoppers* should be put in at and below the floor line; the *dampers* should be opened sufficient to permit of changing the thickness of the fuel bed to suit any changes in the fuel size as well as in the quality of the gas produced. It has been found that a fuel depth of not less than 5 ft. produces good results when the coke from the best coking coals is used as fuel, while a depth of from 2 ft. to 2 ft. 6 in. will answer with coke of a smaller size.

Producers used with modern settings are classified as:

I. Internal producers, constructed inside the setting.

II. External producers, located in close proximity to the setting.

III. Isolated, or often referred to as central producers.

Considerable ground space is saved with *internal* producers, they being located within the bench setting proper, and they are economical in construction; the producer gases have but a short travel to the combustion chamber, and loss of heat is thereby avoided to some extent. These producers require very careful design and construction because, due to their location, they may become a menace to the entire setting by weakening the structure. This type is necessarily rectangular in shape, in consequence of which they have a lower economic efficiency because of the greater wall space as compared with the ideal circular form; besides this, these types possess another disadvantage in that it becomes difficult to maintain a deep as well as level fuel bed, thus permitting possible short circuits of primary air, resulting in the production of gas low in calorific value. This difficulty can be removed to a great extent by providing the producer with a properly designed curtain wall.

The *external* producers, which are located directly in front of the bench, are subject to a greater loss of heat by radiation than the internal ones, but they possess the advantage of being easily and efficiently operated. All parts are accessible, and therefore it is easily possible to prevent the formation of clinker arches, and the fires can be thoroughly and easily cleaned, these conditions leading to a more constant supply of a better quality of producer gas.

The *isolated* producer is built to serve one or more benches, and possesses many advantages not inherent in either of the other types. Less labor and attention are required for their proper operation, the combustion of the fuel is under better control, and the gas produced is of the best quality. Owing to the location of this producer, it is necessary to reheat the gas, but this is readily done by utilizing the heat of the waste gases; the gas from this producer should also be cleaned before it enters the setting, because the clean gas prevents stoppages in the heating flues, thus ensuring a longer life to the setting by preventing a fluxing of the fire-brick material arising from the high-iron content in ash from coke dust drawn into the flues, the fusing point of this ash being considerably below the temperatures maintained in the setting proper.

PROF. KARL BUNTE claims that the use of these isolated, or central producers permits of the utilization of 80 per cent. of the calorific value of the fuel, providing the producer gases are reheated and the radiation losses are low; the almost absolute control of the carbonizing temperatures, due to the regulation of the gas supply by metal valves; and the production of an ash which is practically free from unconsumed carbon, as is the case with the KERPELY producer. Since the producer is supplied with a steam jacket, sufficient steam is generated, by the sensible heat of the producer, to operate the fans and dust removers in addition to that required for gas-making.

The three principal classes of isolated producers are (1) the down-draft producer; (2) the double-zone producer; and (3) the up-draft producer. In the *down-draft producer* the process of gasification is a reverse one, as the gas produced passes down through the fuel bed, the object being to completely split up the tar into fixed gases. This process is accompanied by the formation of large quantities of soot which, while filtering down through the fuel bed, gradually fill up and choke the interstices between the fuel particles. This choking action necessitates an abnormal increase in the pressure of the air blast and causes a rapid deterioration in the quality of the gas.

In order to afford some measure of relief for this excessive fuel-bed resistance, gas-making must be frequently interrupted, during which period a blast of steam, or gas under pressure, is blown through the bed in a reverse direction, it being necessary to draw gas from a storage holder during this interval.

This operation, however, must finally cease, and the fuel bed, after being in service from a week to 10 days, will no longer be capable of satisfactory gas-making; furthermore, to add to these difficulties, the ash, which cannot be removed during the gas-making period in this class of producer, becomes troublesome, and a shut-down therefore becomes imperative. The fuel bed must be cleaned and, in many cases, the fire must be rebuilt, all of which involves much irksome labor and materially increases the cost of attendance. If continuous operation should be desired, a spare unit must be installed for service during this shut-down period. The down-draft producer is restricted to the use of high-grade coal containing a low percentage of ash and sulphur; experience has shown that, when using coal which contains a high percentage of volatile matter, successful commercial operation becomes most difficult.

In this class of producer the fuel bed is not only required to gasify the coal, but it must also act as a gas-filter or soot-separator, and these duties, even when performed separately, present some difficulties; when these duties are combined in one operation, the fuel bed is unduly burdened, thus multiplying the difficulties with the result that neither duty is perfectly discharged.

The *double-zone producer* in its design is essentially a combination of the up-draft and down-draft types, separate air blasts being blown through the machine in both directions, and this double process is subject to many of the difficulties inherent in the down-draft machine. Furthermore, it is very difficult to maintain the distinct combustion zones in the positions intended, this difficulty being materially increased when running hot. "Bridging" of the upper fuel bed also occasions serious interference with the regular downward feeding of the coal; when this obstruction is removed, the fuel descends with a rush, thus making it possible to empty the upper-zone chamber of fuel. A condition such as this disarranges the entire gas-making process, and there is also imminent danger of passing an explosive mixture through the gas mains.

A producer of this type is in no condition to split up the tar, after the machine has been idle for some time, and unless the fuel-bed temperatures are made to respond promptly to variations in the load, the gas as delivered is very apt to become heavily charged with tarry vapors. The complete gasification of all the hydrocarbons, contained in the common grades of American bituminous coals, present many other practical difficulties with this class of producer. Claims have been made that these tar-fixing types of producers leave the gas entirely free from the troublesome tar and soot impurities, but these claims are not always realized in practice, and cleaning apparatus, such as is usually supplied with up-draft machines, must then be provided.

In the *up-draft producer* the function of the fuel bed is restricted solely to gas-making, the cleaning of the gas being a separate and distinct operation. Should it, however, become necessary to decompose the tar, the fuel bed may be run hot, and the soot, instead of lodging in the fuel bed, is then carried away with the gas from which it is removed by a separate cleaning device. A division of these functions, or the separation of the gas-making from the cleaning process, makes this type of producer capable of continuous operation for long periods without letting down for cleaning purposes; besides this, the reliability of the up-draft producer is practically unapproached by any other type, making spare units unnecessary. No large or expensive gas holder is required, because the producer output responds promptly to any sudden increase in demand, and only a small pressure regulator is required to maintain a slight pressure in the mains.

Attempts are being made at present to develop the slagging type of producer in connection with the heating of coke ovens, and a successful issue to this problem will result in a great advance in the application of gaseous-firing, because a single producer can be so constructed as to be able to consume as high as 75 tons of fuel per day with practically continuous operation. Poorer grades of fuel can be used in this isolated type of producer, thus ensuring a greater residual return to the illuminating gas manufacturer, while the use of either of the other types entails charging them with retort coke which has a high market value.

In such localities where there is a good coke market, but where the shipping facilities are such that a favorable market for coal tar does not prevail, the latter has been substituted for coke as bench fuel. The use of tar as producer fuel requires that the ash pan be filled with some igniting material, usually coke breeze, and that the furnace door be bricked up, leaving an opening of approximately 10 sq. in. near the top, thus allowing for the admission of the air required for combustion. This opening also permits the employment of a "V"-shaped iron conduit set at such an angle that the tar, which is usually supplied in a small stream, is caused to fall onto the fired coke breeze in the ash pan; this is the usual method applied when direct-fired benches are operated with tar fuel.

When tar fuel is used with a gaseous-fired setting, the front wall of the producer is sloped down on the inside in order to give a sufficient distilling surface for the tar. The fuel tar should be supplied at constant head, thus ensuring more regular furnace conditions; the tar, dependent on its viscosity, should be first heated, and any foreign matter which might cause stoppages in the lines should be removed by passing the tar through a filter. All supply lines should be provided with valves for properly regulating the supply of tar to each furnace, and they should be located as close to the setting as possible in order to prevent chilling the tar, with a consequent stoppage. Injector burners have been used for supplying tar to the furnace, but a gravity flow answers every purpose, and with the injector burner the amount of steam required for the burners, etc., is from 0.8 to 1.5 times the weight of tar used.

The following example will illustrate the heating value of tar as fuel in the producer, it of course being first necessary to know the composition of the tar, and we will therefore assume a tar having the following constituents:

Constituents	Formula	Per cent.	Weight of combustible	
			C.	H.
Ammoniacal liquor, etc.	H_2O	3	....	....
Light oil, benzol, naphtha, etc.	C_6H_6	5	4.6155	0.3846
Middle oil, creosote oil, etc.	$C_{16}H_{14}$	26	24.2320	1.7680
Heavy oil, anthracene oil, etc.	$C_{14}H_{10}$	4	3.7752	0.2248
Pitch	....	62	51.0800	0.9900
Totals	....	100	83.7027	3.3674

The carbon present in every 100 lb. of tar is, therefore, 83.7027 lb. and the hydrogen is 3.3674 lb.; we will therefore have:

Carbon – 83.7027 × 14,544 = 1,217,372 B.t.u.'s
Hydrogen – 3.3674 × 62,028 = 208,873 B.t.u.'s

Or a total of 1,426,245 B.t.u.'s

giving 14,262 B.t.u.'s per pound of tar.

The amount of CO found in the producer gas is dependent upon operating conditions, such as:

1. The temperature of the producer.
2. Depth of the hot fuel bed.
3. Rate of flow of gas through the bed.

These conditions were investigated by CLEMENT, ADAMS and HASKIN[1] in order to determine the relation existing between the percentage of carbon monoxide formed and the temperature and time of contact, the result of these investigations leading to the statement "in order that the reaction $CO_2 + O = 2CO$ in the fuel bed of the gas producer may be nearly complete and take place in the shortest possible time, a temperature of not less than 1300°C. (2370°F.) must be maintained. The higher the temperature the more rapid will be the formation of carbon monoxide; also since the amount of carbon monoxide formed increases with the time the gases are in contact with the incandescent carbon, the greater the depth of the incandescent portion of the bed, the greater will be the percentage of carbon monoxide obtained."

In the operation of producers we find the temperature rising as high as 1650°C. (3002°F.) before carbon dioxide is eliminated, and at a temperature of 1200°C. (2192°F.) we find from 5 to 10 per cent. of carbon dioxide and from 19 to 28 per cent. of carbon monoxide; in spite of a higher carbon monoxide production, operating conditions limit the producer temperature, and it is very undesirable to make this temperature too high, because if the gases leaving the producer are of too high a temperature they will greatly reduce the efficiency of the process, owing to the sensible heat thus lost. These higher temperatures would also be responsible for the fusion of the ash, with consequent clinker troubles, and the destruction of the firebrick linings. The method used to avoid this trouble is, as explained before, the introduction of steam into the air blast, thus introducing an endothermic reaction. The passage of the steam over the incandescent fuel, or carbon, produces the following reactions:

1. $H_2O + C = H_2 + CO$, and
2. $2H_2O + C = 2H_2 + CO_2$.

Due to these two endothermic reactions, the heat which would otherwise pass out of the producer is absorbed and thus made available for subsequent work.

The decomposition of 1 lb. of water vapor requires 6900 B.t.u.'s and, while the formation of both carbon dioxide and carbon monoxide is exothermic, the dissociation of steam is endothermic, this latter quality causing a temperature reduction of both the fuel bed and the gas. Due to this condition the rate of gas production and the operating temperature of the producer, is under control when a regulated air and steam supply is available.

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[1] Bull. No. 7, Bureau of Mines.

The reaction producing hydrogen and carbon dioxide is the one which governs producer operation to the greatest degree, and the heat absorbed by the steam in the above two reactions is shown in Table XLVIII.

TABLE XLVIII.—ABSORPTION OF HEAT BY STEAM[1]

Reaction	$H_2O + C = H_2 + CO$			$2H_2O + C = 2H_2 + CO_2$		
Relation of weights	18 + 12 = 2 + 28			36 + 12 = 4 + 44		
Process	H_2O, lb.	CO, lb.	B.t.u.	H_2O, lb.	CO, lb.	B.t.u.
Heat absorbed in decomposing steam at 212°F	18	......	−104,580	36	......	−209,160
Heat in steam above water at 70°F	18	......	− 19,944	36	......	− 39,888
Total heat absorbed	18	......	−124,524	36	......	−249,048
Heat developed by the oxidation of C by O of steam	....	12	+ 53,400	....	12	+175,776
Net heat absorption	18	12	− 71,124	36	12	− 73,272
Heat absorption per pound of steam	1	......	− 3,951	1	......	− 2,035
Heat produced per pound of C to CO-($2C + O_2 = 2CO$)	....	1	4,450	1	......	4,450
Allow a total heat loss of 33 per cent. for radiation, ashes, etc.	....	......	1,483	....	......	1,483
Net heat produced	....	......	2,967	....	......	2,967

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[1] R. D. WOOD COMPANY.

The maximum pounds of steam decomposed per pound of carbon burned by air will then be by reaction $H_2O + C = H_2 + CO$

$$\frac{4450 - 1483}{3951} = 0.75, \text{ and}$$

by reaction $2H_2O + C = 2H_2 + CO_2$

$$\frac{4450 - 1483}{2035} = 1.45$$

Under all ordinary conditions it may be stated that there is a definite relation between the amount of steam and air admitted into a producer to secure the maximum operating condition, but this relation must be determined for each individual case. The following test report shows the effect of steam on both the quality and quantity of gas produced.[1]

Producer temperature	1850°F. (1010°C.)	1050°F. (566°C.)
Steam decomposed per pound of carbon	0.5 lb.	0.98 lb.
Air required per pound of carbon	47.5 cu. ft.	47.8 cu. ft.
Volume of gas per pound of carbon	77.2 cu. ft.	87.2 cu. ft.
B.t.u. per cubic foot of gas	176.9	156.8
Combustible in gas	51.5 per cent.	45.5 per cent.
Analysis:		
CO from air	25.7 per cent.	23.0 per cent.
CO from steam	12.9 per cent.	....
H from steam	12.9 per cent.	22.5 per cent.
CO_2 from steam		11.2 per cent.
N from air	48.5 per cent.	43.3 per cent.

It may also be stated that the amount of water vapor which may be decomposed, as well as the composition of the gas evolved, is a function of the temperature and the time of contact, and for any given gas velocity and depth of fuel bed, or "time of contact," the activity of $H_2O + C = H_2 + CO$ becomes greater, and the activity of $2H_2O + C = 2H_2 + CO_2$ becomes less with an increase in temperature. With temperatures below 1100°F. (594°C.) the reaction $2H_2O + C = 2H_2 + CO_2$ becomes active, and the reaction $H_2O + C = H_2 + CO$ does not occur when "time of contact" is determined by practical operating conditions. Intermediate temperatures up to 1700° to 1800°F. (925° to 982°C.) are favorable to the production of carbon monoxide, but at the latter temperature and above, the production of carbon dioxide by the reaction of water vapor practically ceases.

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[1] R. D. Wood Company.

Table XLIX shows the effect of temperature on the reactions between steam and carbon.[1]

TABLE XLIX.—RELATION BETWEEN TEMPERATURE AND STEAM REACTIONS

Temperature		Steam decomposed, per cent.	Composition of gas by volume		
Centigrade	Fahrenheit		CO_2	CO	H_2
364	690	8.8	29.8	4.9	65.2
758	1400	25.3	27.0	7.8	65.2
838	1540	41.0	22.9	15.1	61.9
954	1750	70.2	6.8	39.3	53.3
1010	1850	94.4	1.5	49.7	48.8
1060	1940	96.8	1.3	48.0	50.7
1125	2050	99.4	0.6	48.5	50.9

The general effect of steam admitted into the producer is shown in Table L. The statements tending to explain the presence of methane in small quantities found at times in producer gas, employ the carbon-hydrogen reactions. Recent experiments made by VIGNON indicate that this constituent is more or less dependent upon the time of contact with the fuel. Under ordinary operating conditions these reactions need not be taken into account, as the quantity is usually negligible; this would also apply, owing to the lack of definite knowledge, to those higher hydrocarbons resulting from the decomposition of fuel high in volatile matter, as would apply to producers operating on bituminous coal.

TABLE L.—EFFECT OF STEAM IN PRODUCER

Gas	Excess of steam		
	Moderate	Great	Maximum
Temperature, °F	1472	1292	932
Temperature, °C	800	700	500
Carbon monoxide, CO, per cent	23.5	16.4	11.5
Carbon dioxide, CO_2, per cent	5.3	8.9	15.0
Methane, CH_4, per cent	3.3	2.5	1.9
Hydrogen, H_2, per cent	13.1	18.6	24.6
B.t.u. per cubic foot	151	135	129

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[1] CARPENTER and DIEDERICHS.

There are a number of good isolated producers on the market, but it will be sufficient to describe one type, and the Kerpely producer, which has met with remarkable success in Europe, has been selected for this purpose.

A successful producer plant must necessarily embody the following advantages:

(1) The gas must be free from dust, in order to prevent the deposition of dust in the heat flues.

(2) It should be able to operate on a cheap grade of fuel.

(3) The handling of fuel and ash should involve a minimum of labor.

(4) The gas should be of a uniform quality.

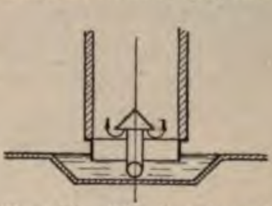

Fig. 58.—Fixed grate and water trough.

The freedom of dust in the gas can be readily secured by washing the producer gas in tower scrubbers, disintegrators, or washers of the Feld type, thus reducing the possibility of flue stoppages to a minimum. Operation with a cheap grade of fuel can be secured with some producers without any difficulty, this condition being exemplified in the Kerpely, where a revolving grate has successfully overcome all ash troubles.

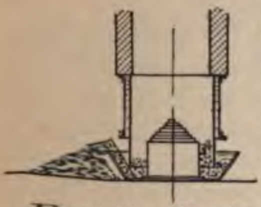

Fig. 59.—Kerpely revolving water trough.

Producers provided with ash pits in the shape of a water trough, the producer shell being supplied with a seal plate dipping into the water, Fig. 58, are favorite designs with many builders of fixed-grate machines, but the Kerpely producer was the first one supplied with a water trough which is caused to rotate on a central axis; originally the outer path of the trough was supplied with rollers to permit of revolving the former, but this method was soon discarded in favor of balls running in a ball race. The advantage claimed for the revolving water trough is due to the fact that in this manner the ash from the producer is projected against a fixed "scraper," and the ash is thus automatically discharged over the side of the trough, this combination of revolving trough and fixed

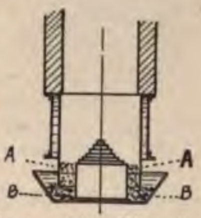

Fig. 60.—Kerpely grate.

scraper being one of the most valuable features in the Kerpely machine.

The Kerpely ash scraper is made to conform to the shape the trough, the angle of inclination of the scraper being such a to effect a most efficient discharge of ash; the scraper does no only push the ash aside, but it also causes the ash to pile up, Fig

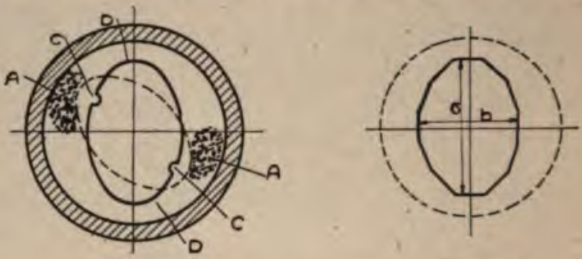

Fig. 61. Fig. 62.

Figs. 61 and 62.—Kerpely eccentric grate.

60, until it reaches the height of the discharge. The constructors of this producer soon followed up the successful operation of the above ash-handling method with a new design of grate, the latter being placed eccentrically upon the revolving water trough, thus causing it to rotate with the latter, with the result that the automatic discharge of ash and clinker into the water trough was ensured, in this manner securing more complete control of the fuel bed.

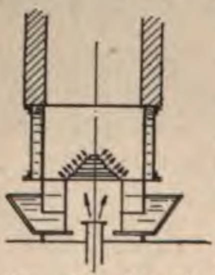

Fig. 63.—Kerpely blast discharge.

This method of operation is readily understood by an examination of Figs. 60 to 63. It will be noted that the grate is oblong in plan and polygonal in shape, being fixed eccentrically on the water trough in such manner that the rotating water trough will produce a continuous crushing effect between the grate and the water seal apron, this action being so gradual and effective that clinker is automatically crushed and then discharged into the water. By referring to Figs. 60 to 62 it will be seen that the clinker mass (*A*), when it enters the ash zone (*B*), is caught by the numerous edges of the revolving grate in such manner that the corners (*C*) press the clinker and ash in a horizontal direction against the seal apron which, with the vertical drop induced at the parts marked (*D*), breaks and crushes the material into small pieces, the latter being then removed from the water trough by means of the scraper.

Fig. 62 shows the method in which the fuel bed is controlled; this oblong and polygonal shaped grate is placed eccentrically upon the rotating trough, it being built up in a tower-like form, and in this manner the air blast and steam supply are brought into intimate contact with the entire area of the fuel bed, which is wider in surface than with the usual grate fixed at the center, the difference in the length of the dimensions (*a*) and (*b*) adding to this effect. Fig. 63 shows the built-up grate section and the method of distributing air blast and steam.

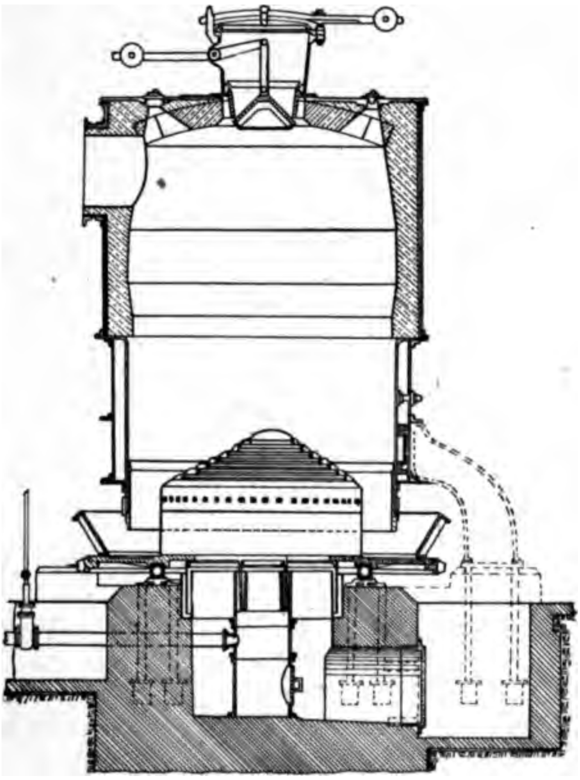

FIG. 64.—Section through Kerpely producer.

The air and steam supply are provided with individual controls, and are distributed through the duplex supply to the central and outer portions of the grate, this construction securing uniform combustion over the entire grate area; the control is arranged that the supply of air and steam to the central and ou portions of the grate can be so adjusted as to ensure the ou portion of the grate not burning too hot in comparison with central portion, or "*vice versa*."

Fig. 64 is a section through the Kerpely producer; the revo ing grate, as here shown, consists of a single oblong and sph cally shaped cone fixed eccentrically upon the water trough provided with a nearly flat, oblong-shaped top. This cone is b up of a number of plates provided with a series of holes, an is claimed that the spaces between these holes are never cho up with ashes, thus permitting the air and steam to reach producer. The rotation of the grate disperses the air and ste over the entire fuel bed, and the nearly flat top of the grate ke the carbon dioxide zone as low as possible.

The producer is also supplied with a water-cooled jac thus reducing clinker troubles to a minimum, as clinkers do cling to water-cooled plates; the cooling water is circulated c tinuously, and it is claimed that the heat absorbed by this wa amounts to a very small percentage of the heating value of fuel gasified. This hot water can be used for boiler feed, or other purpose.

At one plant where these producers are installed, and af an operating period of 3 years, the average analysis of the produced during this time showed:

	Per cent.
CO_2	3.8
CO	27.2
H	9.7
CH_4	3.1
Total combustible	40.0

Average moisture 29 grams per cubic meter. 12.6 grains cubic foot.

Blast saturated with live steam at 35°C. (95°F.).

Comparison tests made with a Kerpely producer and one of dinary fixed-grate design, using inferior coal in medium-sized n of about 2-in. cube down to dust, the analysis of the coal giv

	Per cent.
Fixed carbon	42.03
Vol. hydrocarbons	23.65
Sulphur	2.42
Ash	23.90
Moisture	8.00

showed some remarkable results. Five analyses of the gas produced, made on successive days are shown in Table LI, the last column of this table showing the average gas obtained from the same fuel in a fixed grate producer.

TABLE LI.—COMPARISON BETWEEN KERPELY PRODUCER GAS AND ORDINARY FIXED GRATE PRODUCER GAS

Analyses of gas	1st day	2nd day	3rd day	4th day	5th day	Average gas from same coal, fixed grate producer
Carbon dioxide, per cent	2.8	3.6	3.2	2.0	3.1	8 to 14
Carbon monoxide, per cent	26.8	23.2	25.6	24.4	27.1	15 to 20
Hydrogen, per cent	8.8	9.6	9.6	8.0	8.3	8 to 10
Methane, per cent	3.6	3.2	3.2	3.2	3.1	2.5 to 3
Nitrogen, per cent	58.0	60.4	58.4	62.4	58.4	....
Combustible gases, per cent	39.2	36.0	38.4	35.6	38.5	32
Moisture in grams per cb. m.	29.0	39.0	35.0	34.0	32.0	45 to 48
Moisture in grains per cu. ft.	12.6	17	15.3	15.0	14.0	20 to 21

KERPELY PRODUCER TEST AT THE VIENNA GAS WORKS

These tests were made with the use of two kinds of fuel, viz., lump coke, and unscreened coke, known as coke breeze, a mixture consisting largely of coke dust and greasy coal. The producer had an inside diameter of 2 meters (6.56 ft.). The quantity of fuel available for this test amounted to 44,400 kg. of lump coke mixed with 56,100 kg. of coke breeze and 20,640 kg. of screened coke breeze, the total fuel therefore amounting to 121,140 kg. (266,508 lb.) while an hourly consumption of 704 kg. (1549 lb.) was allowed for. The tests were made under a high pressure and with varying quantities of steam.

During the entire test period the fire was found to be regular and no stoking was required, and this in spite of the fact that the heat in the producer, while gasifying lump coke, was so intense that the temperature of the water leaving the jacket was 90°C. (194°F.). The gasification of the lump coke produced a very dirty ash, the latter being mixed with a small amount of carbon, this condition requiring the admission of more steam in order to prevent the formation of clinker.

During the gasification of the mixture of coke and coke breeze the temperature of the producer was lower, and the resultant ash was improved, no realizable carbon being found in it. The

TABLE LII.—RESULTS OF VIENNA TESTS ON KERPELY PRODUCER

Analyses of fuel and gases
Coke used: ash 10 per cent.; moisture 12 per cent. Calorific power 5700 cal.
Average

Date	Capacity		Combustibles	CO_2	CO	H
	Kilos	Pounds				
22 Sept., 1909, day	4,500	9,900	Lump coke	5.00	25.50	8.80
22 Sept., 1909, night	6,000	13,200		3.90	26.40	8.50
23 Sept., 1909, day	6,000	13,200		5.60	25.10	7.80
23 Sept., 1909, night	7,200	15,840		5.60	24.10	9.10
Average capacity per 24 hours	11,850	26,070	Average analysis	5.02	25.30	8.55
24 Sept., 1909, day	7,800	17,160	½ lump coke and ½ coke breeze	3.60	29.20	9.10
24 Sept., 1909, night	8,250	18,150		3.50	28.20	7.90
25 Sept., 1909, day	7,800	17,160		4.30	27.00	8.00
25 Sept., 1909, night	7,500	16,500		4.10	27.80	8.70
				4.00	27.80	8.00
				3.60	28.00	7.50
				4.50	25.80	8.80
				4.30	26.00	8.00
				4.50	25.80	8.80
Average capacity per 24 hours	15,675	34,485	Average analysis	4.04	27.26	8.31
26 Sept., 1909, day	6,300	13,860	⅓ lump coke and ⅔ coke breeze	No analysis on Sunday		
26 Sept., 1909, night	8,400	18,480				
Average capacity per 24 hours	14,700	32,340	Average analysis	....	....	....
27 Sept., 1909, day	8,850	19,470	Coke breeze	3.80	27.40	8.50
27 Sept., 1909, night	9,000	19,800		4.40	25.80	7.80
28 Sept., 1909, day	12,000	26,400		3.30	27.40	7.60
				4.20	26.50	7.90
				3.30	27.60	6.60
				3.40	27.70	8.20
				3.70	27.70	9.10
				3.40	27.60	7.60
				3.10	28.00	8.50
				3.70	26.50	7.30
Average capacity per 24 hours	19,900	43,780	Average analysis	3.63	27.27	7.91
28 Sept., 1909, night	10,320	22,704	Screened coke breeze	3.00	28.30	8.40
29 Sept., 1909, day	10,320	22,704		4.00	25.90	7.20
29 Sept., 1909, night	End of tests			3.50	26.90	9.40
				3.90	25.90	8.70
				2.90	27.30	7.70
Average capacity per 24 hours	20,867	45,908	Average analysis	3.40	26.84	8.28

temperature of the water leaving the jacket was slightly more than 60°C. (140°F.), and notwithstanding the fact that a small amount of steam was required, the analysis of the gas showed considerable improvement. While burning the screened coke breeze the percentage of carbon dioxide was slightly reduced, the heat in the producer was increased, and the resultant ash again had a dirty appearance, while the water leaving the jacket had a temperature of from 65° to 80°C. (149° to 176°F.).

The depth of the fuel bed was kept normal during the entire test, and, after the various tests were completed, it was decided that the gasification of coke breeze would be most advantageous.

The steam was accurately measured, the consumption amounting to 18.02 kg. per 100 kg. of coke (0.18 lb. per pound of coke).

An efficiency test made on the KERPELY producer at the Vienna works on April 1, 1911, covering a period of 24 hours, showed the following results:

Tonnage of coke breeze gasified:

Producer No. 1.— 9,205 kg.	= 20,251 lb.
Producer No. 2.—10,840 kg.	= 23,848 lb.
Producer No. 3.—11,565 kg.	= 25,443 lb.
Total 31,610 kg.	= 69,542 lb.

Analysis of coke used: 11.75 per cent. moisture; 15.80 per cent. ashes.

Water vaporized in the water jackets: 43,061 liters = 10,185 gal.

Average pressure of steam: 5.36 atmos. = 78.8 lb.

Temperature of feed water: 45.9°C. = 114.6°F.

Average temperature of producer gas: 284.4°C. = 543.92°F.

Highest temperature of producer gas: 360°C. = 680°F.

Lowest temperature of producer gas: 133°C. = 271.4°F.

Average analysis of gas: 2.6 per cent. CO_2; 0.4 per cent. O; 26.8 per cent. CO; 6.8 per cent. H_2; 63.4 per cent. N.

Average analysis of coke: 0.9 per cent. S; 0.7 per cent. N; 0.3 per cent. H_2; 1.0 per cent. O.

Calculated calorific power: 5636 calories.

Total calorific power of coke gasified: 178,153,960 calories.

Production of steam required: 26,247,402 calories.

Calorific power of gas: 981 calories per cubic meter.

Gas produced from 31,610 kg. of coke (69,542 lb.) amounted to 139,753 cubic meters (4,935,238 cu. ft.).

Calorific power of gas produced: 137,097,693 calories.

Specific heat of the gas: 0.31.

Lost heat of the gas: 12,321,183 calories.

Gasification required 0.18 kg. of steam per kilogram of coke gasified.

Calories in coke	178,153,960	
Calories in steam	3,739,322	
Total heat supplied		181,893,282 calories

Heat required for steam production	26,247,402	
Calorific power of gas produced	137,097,693	
Heat retained in gas	12,321,183	
Total heat produced		175,666,278 calories

The total efficiency therefore being $\frac{175,666,278}{181,893,282} = 96.57$ per cent.

The total efficiency being divided into:

For steam production, per cent	14.43
Available gas, per cent	75.37
Heat lost in gas, per cent	6.77

the lost heat being due to cooling the gas down to atmospheric temperature for cleaning purposes.

From this data it is readily seen that the actual effective efficiency of the producer was 89.80 per cent. The total heating surface of the three boilers in the producer top amounted to 180 square meters (1937.5 sq. ft.) and 9.97 kg. of steam were produced per square meter of heating surface (2 lb. per square foot), and 1.36 kg. of steam were produced by the gasification of 1 kg. of coke, or 1.88 kg. of steam per 1 kg. of coke substance free of ash and moisture.

The producer plant at the Leopoldau works in Vienna, consists of twelve machines, each having an average capacity of 15 tons of coke breeze in 24 hours, a cross-section through this plant being shown in Fig. 65. The dust is removed from the gas by means of water disintegrators[1] located ahead of the exhausters, this treatment preventing the deposition of dust in the heating flues of the settings, thus avoiding damage at these points due to fusion between the dust and firebrick material.

Mr. C. J. Ramsburg[2] in comparing the results secured with "built-in" producers and the Kerpely producers at Vienna, presents some very interesting data. He states that, according to Euchene, the B.t.u.'s necessary to carbonize 1 lb. of coal are:

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[1] See "Cleaning of Blast Furnace Gases," 1914, McGraw-Hill Book Co.

[2] American Gas Institute, 1914. "Carbonization in Bulk," Koppers Ovens.

Total heat required		B.t.u.'s
1843 B.t.u.'s	Flue gas losses	602.5
	Heat in gaseous vapors	330.0
	Sensible heat of coke	443.0
	Radiation	495.0
	Formation of vol. comp. endothermic	36.0
	Decomposition of coal	466.0
	Sensible heat of ash	19.0
	Total	2391.5

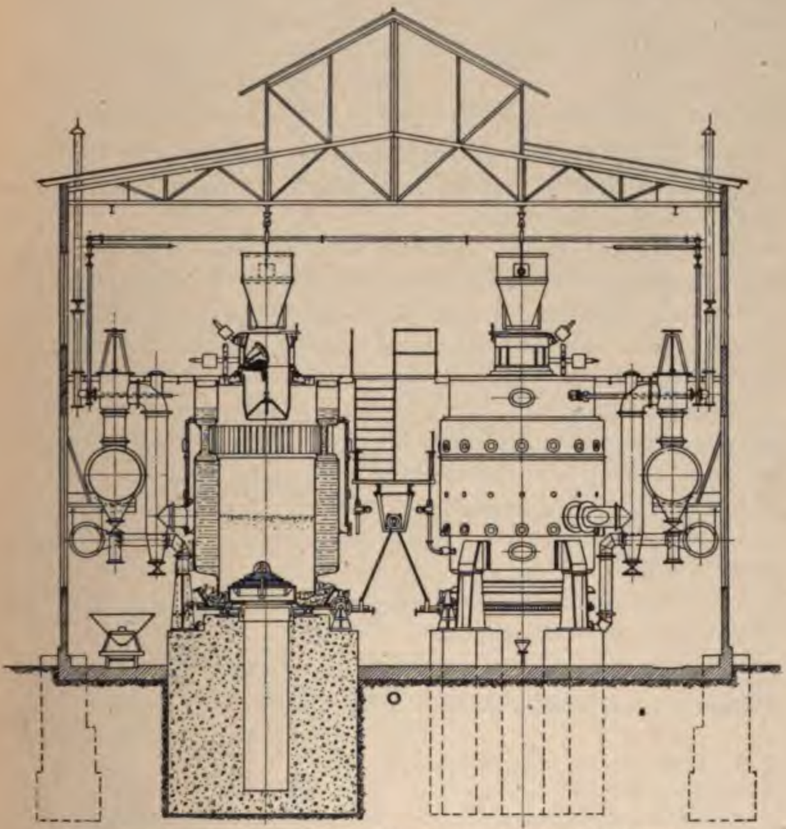

Fig. 65.—Kerpely producer plant at Leopoldau, Vienna.

From this must be deducted 549 B.t.u.'s due to exothermic reactions in the formation of the gases, leaving 1843 B.t.u.'s net. He calls attention to the fact that of this amount only 726 B.t.u.'s per pound of coal pass through the retort, the latter being made up by:

	B.t.u.'s
Gaseous vapors	330
Sensible heat of coke	443
Endothermic reactions	36
Decomposition of coal	466
	1275

From this must be deducted the heat formed in exothermic reactions, leaving a total of 1275 − 549 = 726 B.t.u.'s; the heat lost amounts to:

	B.t.u.'s
Flue gases	603
Radiation	495
Clinker	19
Total	1117

Consequently 1843 B.t.u.'s must be expended in order to secure the benefit of 726 B.t.u.'s, thus showing an efficiency of approximately 39 per cent.

Euchene, as will be explained later, secured these figures on a "stop end bench" which consumed 296 net lb. of coke per net ton of coal carbonized, a high percentage of carbon dioxide being found in the outgoing flue gases, the bench fuel account thus amounting to 14.8 per cent.

Ramsburg also calls attention to the fact that these results were secured under perfect test conditions, the usual operating conditions probably requiring a higher fuel account. As regards the Kerpely producer, operated at the Leopoldau works in Vienna in connection with seventy-two Koppers ovens, divided into blocks of nine chambers each, five blocks having been in use during the testing period, he gives the following from Prof. Bunte's figures:

		Per cent.
Calories in coke	381,230 mill. calories =	97.93
Calories in steam under grate	11,690 mill. calories =	2.97
	392,920 mill. calories =	100.00
The heat returned from this amount was:		
Calories in gas from	312,330 mill. calories =	79.48
Heat contents of the gas	2,032 mill. calories =	0.52
Heat contents of water vapor in gas	7,245 mill. calories =	1.86
Water evaporated	60,969 mill. calories =	15.51
Carbon in ashes	2,512 mill. calories =	0.64
Carbon in flue dust	1,345 mill. calories =	0.34
Total	386,433 mill. calories =	98.35
Unaccounted for	6,487 mill. calories =	1.65
	392,920 mill. calories =	100.00

It is readily seen that from 381,230 million calories introduced into the producer as coke, 312,330 million calories were returned as cold, uniform producer gas ready for the oven battery, all operations having been performed without the necessity of resorting to outside sources for heat or power, and consequently the net efficiency of the producer plant amounted to 81.9 per cent.

The tests showed that in order to carbonize 966,250 kilos (2,130,214 lb.) of coal, 527,525 cubic meters (18,629,000 cu. ft.) of gas were required, the gas having a heating value of 1216 calories per cubic meter (136.7 B.t.u.'s per cubic foot); consequently each kilo of coal required 663 calories in the form of gas (1194 B.t.u.'s per pound of coal).

Using the same 726 B.t.u.'s required for the carbonization of 1 lb. of coal, as given above, RAMSBURG deduces the following:

Heat necessary to carbonize 1 lb. of coal = 726 B.t.u.'s.

Heat necessary in producer gas = 1194 B.t.u.'s.

Heat efficiency of KOPPERS ovens = 60.8 per cent.

Heat in coke necessary to produce 1194 B.t.u.'s in producer gas = $\frac{1194}{0.819}$ = 1458 B.t.u.'s.

Overall heat efficiency of carbonizing system = $\frac{726}{1458}$ = 50 per cent., as against 39 per cent. with horizontal stop-end retorts working with no loss in ash pan or chimney, other than that which pertains to ideal conditions.

Translating this into carbon, he finds that in the retort using 1843 B.t.u.'s per pound of coal, there will be required

$$\frac{1843 \times 2000}{14{,}544} = 254 \text{ lb.}$$

of carbon per ton, or 12.7 lb. per 100 lb. of coal. In contradistinction to this, the producer operated gas oven uses

$$\frac{1458 \times 2000}{14{,}544} = 200.5 \text{ lb.}$$

per ton, or 10 lb. of carbon per 100 lb. of coal.

RAMSBURG then analyzes these two heating systems from the viewpoint of difference in cost of fuel, and assumes that in a retort coal gas plant in America, with a normal gas coal, 1325 lb. of gross coke are secured per net ton of coal carbonized, and that of this amount 300 lb. are used in the built-in producer, leaving

per pound of coal pass through th
up by:

Gaseous vapors............
Sensible heat of coke.........
Endothermic reactions....
Decomposition of coal.

From this must be deduct
reactions, leaving a total of
lost amounts to:

Flue gases.............
Radiation.............
Clinker............
Total......

remainder

Consequently 1843
secure the benefit of 72
approximately 39 per

EUCHENE, as will 1.844
a "stop end bench" —
ton of coal carboniz $1.921
found in the outgo amounts to $2.48 − $1.921 = 56
amounting to 14.8

RAMSBURG also built-in producer required 0.4 lb. of
were secured un pound of combustible, or
conditions pro
gards the KER lb. of steam per ton of coal.
in Vienna
divided into steam is worth 40 cts. per net ton, the steam
been in use further charge of
from PRO

$$\times \frac{40}{2000} = 1.232 \text{ cts.}$$

Calories
Calories cost of the producer gas, without labor,

The he
$1.23 = 57.23$ cts.
Calori
He
Heat coal that 10,500 cu. ft. of coal gas are se-
Wat coal carbonized, the cost of producer fuel per
of gas will be under the above conditions:
Coal

$$\frac{57.23}{10.5} \quad 5.45 \text{ cts.}$$

Referring to the central producer plant, he states that the breeze made will amount to 120 lb. per net ton, and that this will all be used in the producer for fuel. Assuming that 200 lb. of carbon are necessary in the producers per net ton, and that the breeze contains 84 per cent. of carbon and the coke 87 per cent., he finds that, per net ton of coal carbonized, the cost of fuel will be

$$120 \text{ lb. of breeze} = 120 \times \frac{\$1.50}{2000} = 9 \text{ cts.}$$

and

$$115 \text{ lb. of coke} = 115 \times \frac{\$4.00}{2000} = \underline{23 \text{ cts.}}$$

or a total of 32 cts.

Therefore, assuming that all the steam required will be made in the producer plant, the cost of furnace fuel under these conditions will be

$$\frac{32}{10.5} = 3.04 \text{ cts. per 1000 cu. ft.}$$

of gas produced, a saving in the fuel account of 5.45 − 3.04 = 2.41 cts. per 1000 cu. ft. of gas in favor of the central producer plant, this figure being equivalent to a net saving of 57.23 − 32 = 25.23 cts. per net ton of coal carbonized.

The Slagging Type of Gas Producer.—The object of the slagging type of producer is an effort to obtain the advantages and to avoid the disadvantages resulting in the operation of the usual type of producer at high temperatures, the slagging type being one in which the ash is removed in the shape of a liquid slag, it being deemed possible to thus utilize fuel high in ash content, and to maintain a temperature which will be ample for the production of a high percentage of carbon monoxide.

The slagging or blast-furnace type of gas producer is a very early development, the first one having been built by EBELMAN in 1840 at the Audincourt iron works in France, but further development was not attempted until very recently, and in 1907 a battery of these producers was installed at the Gironcourt-on-Vosges glass works, their operation being reported as very satisfactory. It is reported that the amount of heat required to fuse the ash is very small, and that it is much less than the heat value of the carbon lost in the ash from the usual producer. Table LIII gives the analysis of fuel used and gas produced at this plant.

1025 net lb. of coke and breeze for sale; h of this amount 10 per cent. is breeze, thu of breeze and 922 lb. of coke.

Assuming that coke has a value of $ of $1.50 per ton, he finds, with 10 per breeze, that had none of this been used coke and breeze produced would ha

133 lb. of breeze at

1192 lb. of coke at

Total

After deducting the produce is worth:

103 lb. of bre

922 lb. of

Total

Consequently the cost cts. per ton of coal carb

He also states that steam under the grate

254 × 0.4 =

Assuming that th required would add

thus making th amount to

Upon the cured per thousand

and the

are such as
sorb heat in others;
ine almost entirely to
because the greater por-
to carbon monoxide, while
oxide. The heat thus generated

ater vapor, producing hydrogen,
dioxide.

of the fresh fuel, as charged, up to
roducer.

the outer walls of the producer, and
ained in the gas discharged from the

carbon monoxide as is possible should
duction is usually a factor of tem-
therefore, the higher the temperature

Smith.

inside the producer, and the greater the fuel surface exposed to the gases produced, the greater will be the resultant percentage of carbon monoxide. In spite of this theory, it is practically impossible to produce a gas free of carbon dioxide, because certain temperature limits are set, and within these limits the reduction of carbon dioxide to carbon monoxide can only proceed to a given ratio for the temperature. It must also be remembered that the temperature within the producer exercises a very important bearing upon the volatile matter produced if uncarbonized fuels are used, a higher temperature increasing the amount of combustible and, although a smaller quantity of gas per carbon unit is produced, this gas will contain a larger portion of the heat energy of the fuel. The temperature also exercises great influence upon the quantity of condensible products produced, the higher temperatures tending to form soot or pitch, while the lower ones usually produce water and tar. Large fuel surfaces exposed to contact with the ascending gases tend to the formation of a larger percentage of carbon monoxide, and the more porous the fuel, and the greater its depth, the better will be the result, but the depth should never be such as to produce an excessive resistance to the passage of the air or of the gas produced.

The efficiency of the producer is usually expressed as the *Useful Effect* divided by the sum of the *Latent* and *Sensible Heat* supplied. Besides the usual term "*Efficiency*," the expressions "*Cold Gas Efficiency*," and "*Hot Gas Efficiency*" are also used, the former being the result of dividing the usual effect, or *Latent Heat*, by the *Heat Supplied;* and the latter the result of dividing the sum of the *Latent* and *Sensible Heats* by the *Heat Supplied.* In the operation of the producer, a portion of the sensible heat of the gas produced is removed in heating water, in preheating the air blast, etc., and, due to this fact, the real efficiency of the plant will always be greater than the cold gas efficiency.

In working up the heat balance we will assume that the heat of combustion of 1 lb. of carbon, burned to 3.66 lb. of carbon dioxide, amounts to 14,544 B.t.u.'s, and that the heat of combustion of 1 lb. of carbon, burned to 2.33 lb. of carbon monoxide, amounts to 4350 B.t.u.'s; then the heat of combustion of 2.33 lb. of carbon monoxide, burned to 3.66 lb. of carbon dioxide, will be represented by the difference between these two amounts, or 14,544 – 4350 = 10,194 B.t.u.'s. Also, the heat required to dissociate 1 lb. of water vapor will be taken at 6900 B.t.u.'s.

In order to develop a heat equation per pound of carbon gasified in the producer, let

x = the portion of the pound of C burned to CO_2.
$1 - x$ = the portion of the pound of C burned to CO.
y = the pounds of water, in the form of steam, introduced into the producer per pound of carbon gasified.
S = the heat supplied in the steam.
A = the heat supplied in the air.
P = the heat necessary to elevate the initial fuel temperature to the producer temperature.
L = radiation losses.
G = the sensible heat of the gas.

Then,

$$4350(1 - x) + 14{,}544x + S + A = 6900y + G + L + P \quad (40)$$

The heat in the gas per pound of carbon gasified will be:

$$\text{From CO} = 10{,}194(1 - x)$$
$$\text{From } H_2 = 6900y$$

the total heat available being

$$10{,}194(1 - x) + 6900y$$

The "*Cold Gas Efficiency*" will be:

$$\frac{10{,}194(1 - x) + 6900y}{14{,}544 + S + A} \quad (41)$$

and the "*Hot Gas Efficiency*" will be:

$$\frac{10{,}194(1 - x) + 6900y + G}{14{,}544 + S + A} \quad (42)$$

The composition of the producer gas, per pound of carbon gasified, will be:

$$CO = (1 - x) \times \frac{28}{12} = 2.33\ (1 - x) \text{ lb.}$$

$$CO_2 = x \times \frac{44}{12} = 3.66x \text{ lb.}$$

$$H_2 = y \times \frac{2}{18} = \frac{y}{9} \text{ lb.}$$

For the determination of the nitrogen content, the quantity of air required for combustion must first be calculated, as follows:

Oxygen required:

For CO $= (1 - x)\frac{16}{12} = 1.33\ (1 - x)$ lb.

For $CO_2 = x \times \frac{32}{12} = 2.66x$ lb.

The decomposition of water produces $\frac{16}{18}y = \frac{8}{9}y\ O_2$, and the oxygen content of the air supplied will equal

$$1.33\ (1 - x) + 2.66\ x - \frac{8}{9}y = 1.33\ (1 + x) - \frac{8}{9}y$$

The air necessary per pound of carbon will be

$$\left\{1.33(1 + x) - \frac{8}{y}y\right\} \times \frac{100}{23.5}$$

and its oxygen equivalent will be

$$\left\{1.33(1 + x) - \frac{8}{9}y\right\} \times \frac{76.5}{23.5}\ \text{lb. of N}$$

Consequently a producer gas of ideal composition, containing carbon dioxide, carbon monoxide, hydrogen, and nitrogen in proper proportion per pound of carbon gasified, will weigh

$$2.33(1 - x) + 3.66x + \frac{y}{9} + 1.33\ (1 + x) - \frac{8}{9}y + \left\{1.33(1 + x) - \frac{8}{9}y\right\}\frac{76.5}{23.5} = 8.01 + 7.01\ x = 3.67y\ \text{lb.}$$

and the volumetric composition per pound of carbon gasified, assuming the following unit weights, will be:

CO = 0.07807 lb. per cubic foot.
CO_2 = 0.12267 lb. per cubic foot.
H_2 = 0.00559 lb. per cubic foot.
N_2 = 0.07831 lb. per cubic foot.

Volume of CO:

$$\frac{2.33}{0.07807}(1 - x) = 29.84\ (1 - x)\ \text{cu. ft.}$$

Volume of CO_2:

$$\frac{3.66x}{0.12267} = 29.02x\ \text{cu. ft.}$$

Volume of H_2:

$$\frac{y}{9 \times 0.00559} = 19.87y\ \text{cu. ft.}$$

Volume of N_2:

$$(55.51 + 41.74x - 37.1y)\ \text{cu. ft.}$$

[illegible] the volume of producer gas per unit [illegible] gasified is equivalent to

$$[illegible] - 40.92x - 17.23y$$

[illegible] under ideal conditions, in which $x = 0$, [illegible] of the total sensible heat, or $G = S + A$, [illegible] above figures in the general heat equation, or [illegible] and assuming $P = 360$ B.t.u., we will have

$$\frac{360}{[illegible]} = 0.578 \text{ lb. hydrogen.}$$

[illegible] cu. ft.

[illegible] × 0.578 = 11.485 cu. ft.

[illegible] cu. ft.

[illegible] production of 74.575 cu. ft. of gas per pound [illegible].

[illegible] analysis of this gas gives:

[illegible] per cent.

[illegible] per cent.

[illegible] per cent.

[illegible] value of this gas would be:

[illegible]	29.84 × 4350 × 0.07807 =	10,134 B.t.u.
[illegible]	11.485 × 62.028 × 0.00559 =	3,982 B.t.u.
[illegible] per pound of carbon gasified		14,116 B.t.u.

[illegible] calorific value per cubic foot will be

$$\frac{14{,}116}{74.575} = 189.3 \text{ B.t.u.}$$

CHAPTER VIII

FLUE GASES

The proper regulation of furnace, or producer conditions is evidenced in the constitution of the waste gases, because, if perfect combustion of the fuel is secured, the resultant waste gas will contain carbon dioxide and nitrogen, but no carbon monoxide. This condition requires that frequent analyses of the flue gases be made in order to determine whether the furnace conditions are what they should be or not, and in this manner to be able to regulate the admission of both air and steam to the producer in an attempt to secure conditions as near ideal as possible.

Under ordinary conditions of analysis, the relative proportions of carbon dioxide, carbon monoxide, and oxygen in the flue gases only are determined, the amount of nitrogen present being determined by calculation. As was explained before, 1 lb. of carbon requires 2.666 lb. of oxygen, or a volume of 32 cu. ft., at 60°F., for its complete combustion, the resultant product being carbon dioxide which, when cooled, will amount to exactly 32 cu. ft., or the same as the oxygen. If, however, air is used instead of pure oxygen, thus carrying in nitrogen in the proportion of 20.91 O to 79.09 N, the resultant volume of carbon dioxide after combustion will be the same as that of the oxygen, plus a further increase in volume due to the nitrogen.

With complete combustion, using air instead of oxygen, the resultant flue gas would consist of 20.91 per cent. of carbon dioxide, and 79.09 per cent. of nitrogen, but if more air should be used than is required to give the requisite amount of oxygen, we find that the combined volume of carbon dioxide and oxygen found in the flue gas will be the same as that of the oxygen before combustion; therefore, in order to secure complete combustion of the carbon the sum of the percentages by volume of the carbon dioxide and oxygen in the flue gas will always amount to 20.91, even though an excess of air be admitted to the furnace. Carbon monoxide, which can only be produced by an imperfect combustion of the carbon, will occupy twice the volume of the oxygen entering into its composition, and therefore the volume of the flue gases will be greater than that of the air supply in the proportion of $\frac{100}{100 - 0.5 \text{ the per cent. of CO}}$; this condition leads to the statement that when pure carbon is consumed in a furnace, the sum of the percentages of carbon dioxide, oxygen, and one-half of the carbon monoxide must be in the same ratio to the nitrogen as is oxygen in air, or as 20.91 to 79.09.

If the coal contains hydrogen, its action will be to increase the apparent percentage of nitrogen in the flue gases; this is due to the fact that the water vapor will condense at such temperatures at which the flue-gas analysis is usually made, its presence, as a rule, being thereby undetected except in so far as the nitrogen, which accompanied the oxygen, and which combined with the hydrogen, maintains its gaseous form and passes on with the other gases into the analyzing apparatus.

As an example, we will assume an analysis of flue gas to show 12.5 per cent. of carbon dioxide, 0.6 per cent. of carbon monoxide, and 6.5 per cent. of oxygen, all by volume. The nitrogen, which is the only other constituent of flue gas usually considered, will represent a percentage of the total volume of

$$100 - (12.5 + 0.6 + 6.5) = 80.4 \text{ per cent.}$$

We will assume the unit volume designated in this example as 100 per cent. to represent 100 cu. ft., and, with the data given in Table LIV, we find the weight of this flue gas per unit volume of 100 cu. ft. to be,

Carbon dioxide = 0.12269 × 12.5 = 1.5336 lb.
Carbon monoxide = 0.07807 × 0.6 = 0.0468 lb.
Oxygen = 0.08922 × 6.5 = 0.5799 lb.
Nitrogen = 0.07829 × 80.4 = 6.2945 lb.

Total 100.0 = 8.4548 lb.

Due to the combining weights of the elements, it is known that in a unit of carbon dioxide the oxygen constitutes eight-elevenths of the weight, the remaining three-elevenths being carbon; and in carbon monoxide four-sevenths is oxygen by weight, and three-sevenths carbon. Therefore, the weight of oxygen in the 100 cu. ft. of flue gas in this example is:

Oxygen in CO_2.................... $1.5336 \times 8/11 = 1.1154$ lb.
Oxygen in CO...................... $0.0468 \times 4/7 = 0.0267$ lb.
Free oxygen........................ = 0.5799 lb.

Total weight of oxygen........... = 1.7220 lb.

Table LIV.—Density of Gases at Atmospheric Pressure[1]

Gas	Symbol	Specific gravity, air = 1	Weight 1 cu. ft. at 32°F., pounds	Cubic feet per pound at 32°F.	Relative density, hydrogen = 1	
					Exact relative densities	Approximate whole numbers
Oxygen	O	1.053	0.08922	11.208	15.87	16
Nitrogen	N	0.9673	0.07829	12.773	13.92	14
Hydrogen	H	0.0696	0.005621	177.90	1.00	1
Carbon dioxide	CO_2	1.5291	0.12269	8.151	21.83	22
Carbon monoxide	CO	0.9672	0.07807	12.809	13.89	14
Methane	CH_4	0.5576	0.04470	22.371	7.95	8
Ethane	C_2H_6	1.075	0.08379	11.935	14.91	15
Acetylene	C_2H_2	0.920	0.07254	13.785	12.91	13
Sulphur dioxide	SO_2	2.2639	0.17862	5.598	31.96	32
Air	......	1.0000	0.08071	12.390	....	....

[1] Adapted from Kent. (Stirling, 1906.)

The weight of carbon, determined from the same gas analysis, is:

Carbon in CO_2	$1.5336 \times 3/11$	= 0.4183 lb.
Carbon in CO	$0.0468 \times 3/7$	= 0.0201 lb.
Total weight of carbon		= 0.4384 lb.

The atmospheric air supplied to the furnace contains 23.15 per cent. of oxygen by weight, therefore the weight of air which contains 1.7220 lb. of oxygen is $\frac{1.7220 \times 100}{23.15} = 7.439$ lb.; as this amount of air was required for the combustion of 0.4384 lb. of carbon, the weight of air per pound of carbon is $\frac{7.934}{0.4384} = 16.97$ lb.

If the coal used in this example contained 86 per cent. of carbon, 4 per cent. of hydrogen, and 2.5 per cent. of oxygen, the air per pound of coal will have been 16.97 × 0.86 = 14.59 lb., disregarding the hydrogen and oxygen; but the oxygen in the fuel renders one-eighth of its weight of hydrogen inert; therefore only the remainder of the hydrogen is available for combustion, in consequence of which the air required to burn the hydrogen is:

$$\left(0.04 - \frac{0.025}{8}\right)34.62 = 0.0369 \times 34.62 = 1.277 \text{ lb.}$$

34.62 being the weight of air required for the combustion of 1 lb.

of hydrogen, and the total air supply per pound of this fuel is $14.59 + 1.28 = 15.87$ lb.

Knowing the ultimate analysis of the fuel used, the air required for complete combustion, allowing for an excess, can be determined from the following approximate equation, or pounds of air required per pound of fuel =

$$34.62\left(\frac{\mathrm{C}}{3} + \mathrm{H} - \frac{\mathrm{O}}{8}\right) \qquad (43)$$

in which C, H, and O represent the percentage by weight of carbon, hydrogen, and oxygen in the fuel divided by 100.

Knowing the flue-gas analysis, the total amount of air supplied can be found from pounds of air supplied per pound of fuel =

$$3.032\left(\frac{\mathrm{N}}{\mathrm{CO_2} + \mathrm{CO}}\right) \times \mathrm{C} \qquad (44)$$

in which N, CO, and CO_2 represent the percentage by volume of nitrogen, carbon monoxide, and carbon dioxide in the flue gas, while C is the proportionate part, by weight, of the carbon in the fuel.

The weight of the flue gas will be 1 minus the per cent. of ash, expressed in hundredths, more than the weight of air given above, or it will be the sum of the weights of the air and the combustible and moisture in the fuel, or the weight of flue gas per pound of fuel =

$$3.032\left(\frac{\mathrm{N}}{\mathrm{CO_2} + \mathrm{CO}}\right) \times \mathrm{C} + (1 - A) \qquad (45)$$

in which A represents the proportionate part, by weight, of the ash in the fuel.

The ratio of the air actually supplied per pound of carbon to that theoretically required to burn it will be given by

$$3.032\frac{\mathrm{N}}{\mathrm{CO_2} + \mathrm{CO}} + 11.52 = 0.2632\frac{\mathrm{N}}{\mathrm{CO_2} + \mathrm{CO}} \qquad (46)$$

in which N, CO_2, and CO represent the percentages by volume of nitrogen, carbon dioxide, and carbon monoxide in the flue gas.

The ratio of the air supplied per pound of fuel to the amount theoretically required is given by

$$\frac{\mathrm{N}}{\mathrm{N} - 3.782\,\mathrm{O}} \qquad (47)$$

which equation is derived as follows:

The nitrogen in the flue gas is the content in the whole amount of air furnished, while the oxygen in the flue gas is due to the air supplied and not used. This oxygen was accompanied by 3.782 times its volume of nitrogen, and N − 3.782 O represents the nitrogen content in the air actually required for combustion; therefore, N ÷ (N − 3.782 O) is the ratio of the air supplied to that required. The per cent. of excess air is this ratio minus 1.

TABLE LV.[1]—RATIO OF TOTAL AIR SUPPLIED TO THAT THEORETICALLY REQUIRED FOR VARIOUS ANALYSES OF FLUE GAS

$$\text{Ratio} = \frac{N}{N - 3.782O}$$

$CO_2 + CO$	N = 79 CO_2+CO $+O = 21$	N = 79.5 CO_2+CO $+O = 20.5$	N = 80 CO_2+CO $+O = 20$	N = 80.5 CO_2+CO $+O = 19.5$	N = 81 CO_2+CO $+O = 19$	N = 81.5 CO_2+CO $+O = 18.5$	N = 82 CO_2+CO $+O = 18$
21	1.00	....	....	....	....	....	....
20	1.05	1.02	1.00	....	....	....	....
19	1.11	1.08	1.05	1.02	1.00	....	....
18	1.17	1.14	1.10	1.08	1.05	1.02	1.00
17	1.24	1.20	1.17	1.13	1.10	1.07	1.05
16	1.32	1.27	1.23	1.20	1.16	1.13	1.10
15	1.40	1.35	1.31	1.27	1.23	1.19	1.16
14	1.51	1.45	1.39	1.35	1.30	1.26	1.23
13	1.62	1.55	1.50	1.44	1.39	1.34	1.30
12	1.76	1.68	1.61	1.54	1.49	1.43	1.38
11	1.92	1.82	1.74	1.66	1.60	1.53	1.48
10	2.11	2.00	1.90	1.81	1.72	1.65	1.59
9	2.35	2.21	2.08	1.97	1.88	1.79	1.71
8	2.65	2.47	2.31	2.18	2.06	1.95	1.86
7	3.03	2.80	2.59	2.44	2.27	2.14	2.03
6	3.55	3.22	2.96	2.74	2.54	2.38	2.24
5	4.27	3.81	3.44	3.14	2.89	2.68	2.50
4	5.37	4.65	4.11	3.68	3.34	3.05	2.83
3	7.23	5.97	5.10	4.45	3.96	3.56	3.25
2	11.06	8.34	6.71	5.63	4.85	4.27	3.82
1	23.51	13.83	9.83	7.64	6.27	6.12	4.64

The heat lost in the flue gases can be found by the aid of equation (48), or

$$L = 0.24W(T - t) \tag{48}$$

in which L = B.t.u. lost per pound of fuel.
W = weight of the flue gases in pounds per pound of fuel.
T = temperature of the flue gas, °F.
t = temperature of the atmosphere, °F.
0.24 = specific heat of the flue gas.

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[1] STIRLING, 1906.

The heat lost in the carbon monoxide in B.t.u.'s per pou fuel is given by equation (49), or

$$L' = 10{,}150 \times \left(\frac{\mathrm{CO}}{\mathrm{CO} + \mathrm{CO_2}}\right) \times \mathrm{C}$$

in which CO and CO_2 again represent the per cent. by volur carbon monoxide and carbon dioxide in the flue gas, and C proportion, by weight, of the carbon in the fuel.

Diagram III is a graphic representation[1] of the loss of heat ried away by dry chimney gases for varying percentages of ca dioxide, and different temperatures of exit gases, while Dia IV[1] graphically represents the heat lost due to such carbon i

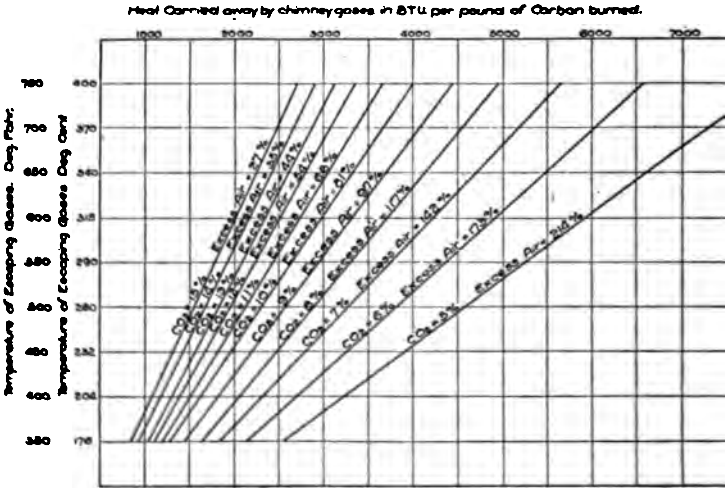

DIAGRAM III.—Heat lost in chimney gases for varying percentages c bon dioxide based on boiler room temperature of 80°F.—N in flue 80.5%—CO in flue gas at 0%.

fuel as is not completely consumed, but which escapes up chimney in the form of carbon monoxide.

The usual analysis of a flue gas is generally secured by mea the Orsat apparatus, shown in Fig. 66. The burette (A graduated in cubic centimeters up to 100, and it is surrounde a water jacket for the purpose of preventing any change in perature which might affect the density of the gas underg analysis. The most accurate analysis is secured with the u four pipettes, marked (B), (C), (D), and (E), the first of w contains a solution of caustic potash for the absorption of carbon dioxide; the second contains an alkaline solution of pyrogallol for the absorption of oxygen; and the remaining two contain an acid solution of cuprous chloride for the absorption of carbon monoxide. Each of these pipettes contains a number of glass tubes, so arranged as to cause some of the solution to adhere to them, in this manner facilitating the absorption of the gas. The pipettes (*D*) and (*E*) contain some copper wire for the purpose of re-energizing the solution as it becomes weakened. A rubber bag is fitted to the rear half of each pipette, this acting to protect the solution from the action of the air; one of these rubber bags is shown at (*K*), Fig. 66.

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[1] Steam, Babcock and Wilcox, Company, 1913.

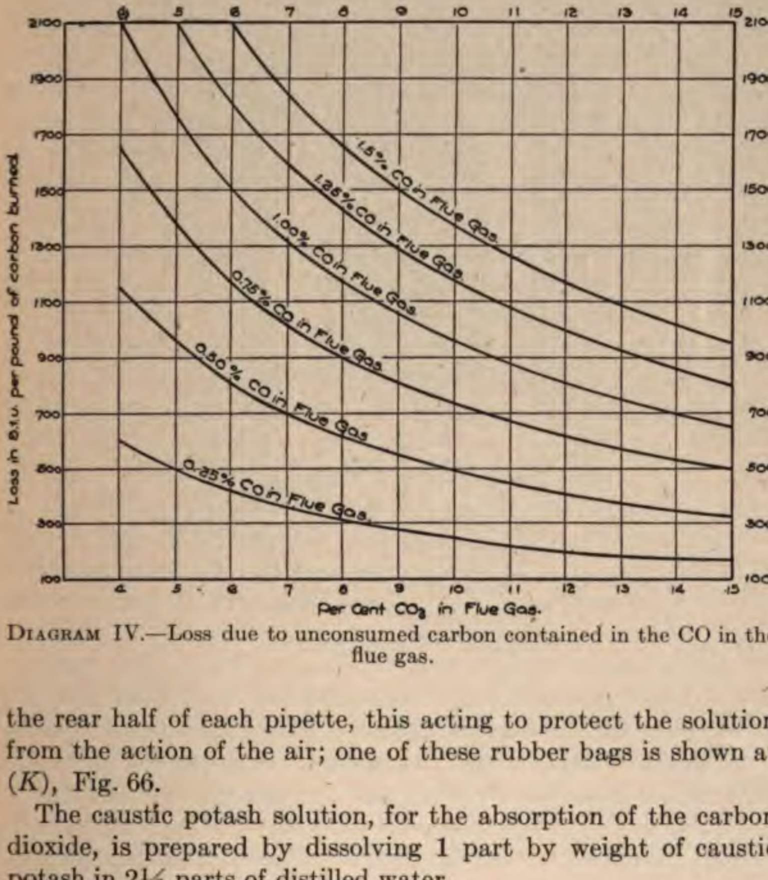

DIAGRAM IV.—Loss due to unconsumed carbon contained in the CO in the flue gas.

The caustic potash solution, for the absorption of the carbon dioxide, is prepared by dissolving 1 part by weight of caustic potash in 2½ parts of distilled water.

The alkaline solution of pyrogallol, for the absorption of oxygen, is prepared by dissolving 1 part by weight of pyrogallic acid in 2 parts of hot water, to which must be added 3 parts of the [illegible] being prepared as d

[illegible] the absorption of the
[illegible] 1 part by weight of
[illegible]oric acid, to which 2
[illegible] added, the solution bei
[illegible] Before using this solution,

[illegible] the proper solution in each
[illegible] mark on the capillary tube

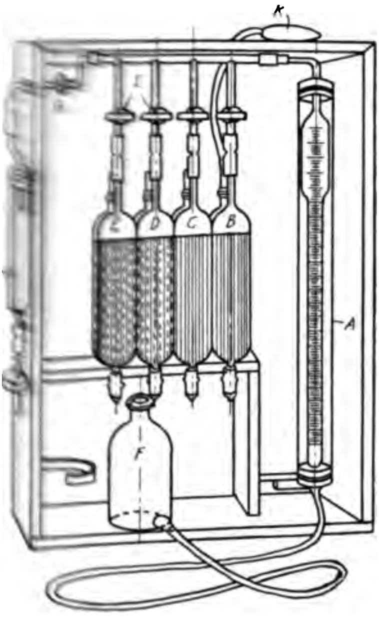

Fig. 66.—Orsat apparatus.

[illegible] drawn into the burette through the U-tube (
[illegible] filled with spun glass, or a similar material,
[illegible] cleaning the gas. In order to discharge any a
[illegible] the cock (G) should be opened to the air
[illegible] raised until the water flowing into the
[illegible] 100 cc. mark; the cock (G) is now turned for
[illegible] the air opening and thus allowing the g
[illegible] the U tube (H), the water bottle (F) being
[illegible] Gas is allowed to enter the burette to
[illegible] mark, after which cock (G) is again opene

air and the excess gas expelled until the water level in both the burette (A) and water bottle (F) is at the zero mark, thus enabling the operator to secure a zero reading at atmospheric pressure. Accurate analysis requires that the apparatus should be absolutely tight and it should be tested for leakage in all its parts, including the connections. As an example of testing for leakage, it may be stated that if after the cock (G) is closed and the water bottle (F) is placed on top of the frame for a short time, thus allowing water to flow into the burette, and then lowered to bring the water to the zero mark, a leak will be indicated if the water in the burette (A) stands above the zero mark.

Before a final sample is drawn for analysis, the burette (A) should be filled with gas and emptied several times, this precaution being necessary in order to make sure that the apparatus will be filled with new gas during the test. The cock (G) is now closed and the cock (I) in the pipette (B) is opened, the gas being then forced over into (B) by raising the water bottle (F). The gas is drawn back into (A) by simply lowering the bottle (F), and when the solution in (B) has reached the fixed mark on the capillary tube, the cock (I) should be closed and a reading taken on the burette, taking care to bring the level of the water in the bottle (F) to the same level as the water in the burette (A). This operation must be repeated until a constant reading is obtained, the number of cubic centimeters read off the burette giving the percentage of carbon dioxide in the gas.

After this reading has been taken, the gas is forced over into the pipette (C), where an operation similar to that described above is performed. The difference between this second reading and the first reading will then give the percentage of oxygen in the gas.

The final operation requires that the gas be driven over into pipette (D), in the same manner as described above, this gas being given a final washing in (E), after which it is passed into pipette (C) in order to neutralize any hydrochloric acid fumes which may have been given off by the cuprous chloride solution, as the latter, if it be an old solution, is prone to give off these fumes, in this manner increasing the gas volume, thus making the burette reading less than the true amount.

The entire process of analysis must be performed in the manner described, because the pyrogallol solution will also absorb carbon dioxide, and the cuprous chloride solution will also absorb oxygen. Due to the fact that the gas pressure in the flue is less than that of the atmosphere, the flue gas will not of itself flow through the pipe connecting the flue to the apparatus, and it is therefore necessary that this flow be aided. This may be done in several ways; the gas may be caused to flow through the pipe in the manner described for filling the apparatus, but this method is a very tedious one. In order to secure quicker results, a rubber bulb aspirator should be connected to the air outlet of the cock (*G*), by means of which a new supply of gas can readily be drawn into the pipe, after which the apparatus can be filled in the manner described above.

The analysis secured by this apparatus is volumetric, and if an analysis by weight should be desired, it can be secured by multiplying the percentages by volume by the molecular weight of each gas, and dividing the products thus secured by the sum of all the products; the quotients will be the percentages by weight. The molecular weights of the gases ordinarily considered are:

Carbon dioxide	44
Carbon monoxide	28
Oxygen	32
Nitrogen	28

Table LVI indicates the method used in converting a volumetric flue gas analysis into one by weight.

TABLE LVI.—CONVERSION OF A FLUE-GAS ANALYSIS BY VOLUME TO ONE BY WEIGHT

Gas	Analysis by volume, per cent.	Molecular weight	Volume times molecular weight	Analysis by weight, per cent.
Carbon dioxide	12.2	$12 + (2 \times 16)$	536.8	$\frac{536.8}{3022.8} = 17.7$
Carbon monoxide	0.4	$12 + 16$	11.2	$\frac{11.2}{3022.8} = 0.4$
Oxygen	6.9	2×16	220.8	$\frac{220.8}{3022.8} = 7.3$
Nitrogen	80.5	2×14	2254.0	$\frac{2254.0}{3022.8} = 74.6$
Total	100.0		3022.8	100.0

Under ordinary conditions, a high percentage of carbon dioxide in the flue gas is taken as indicative of good combustion with a consequent high efficiency, but this is only true in that a high carbon dioxide reading indicates the use of the small amount of excess air which usually accompanies good combustion; therefore a high percentage of carbon dioxide alone does not make an Orsat reading entirely reliable. An increased percentage of carbon dioxide in the flue gas is accompanied by a tendency toward the presence of carbon monoxide and it is very hard at times to detect the latter with an Orsat apparatus. This is especially true if the cuprous chloride solution is not properly prepared, and it must be fresh and capable of absorbing carbon monoxide. A small amount of carbon monoxide, say 1 per cent., in the flue gas will not have much influence on the quantity of excess air, but the presence of even this small amount will indicate a loss due to incomplete combustion of the carbon in the fuel which may amount to 4.5 per cent. of the total heat in the fuel burned.

The ordinary methods of taking samples of flue gas usually lead to inaccurate results in analyses, because the gas mixture is not always homogeneous. It is not a difficult problem to obtain a sample of gas from a given point in the furnace or flue, but as a rule the composition of the gas always changes, and therefore it becomes somewhat difficult with the usual methods used to secure an average sample. Consequently, it becomes necessary to use an apparatus for collecting the gas which will represent an average composition at the point of collection during a desired period, or the sample must be collected instantaneously, so that it will represent the composition of the gas at the point selected and at the instant of collection.

FRAZER and HOFFMAN[1] state that a sample thus taken is only representative of the gas occupying a certain space surrounding the point of collection, and that in order to determine the average composition of the entire volume of gas it becomes necessary to multiply the number of samples and to distribute the points of collection in such manner that the average of the samples will correctly represent the entire gas body. The number of samples taken should depend on the differences in composition that are presumed to exist throughout the volume of gas to be sampled, and that the difference in composition between the samples taken at any two adjacent points of collection should not be greater than from 0.3 to 0.5 per cent.

These authors state that the sampling of flue gases can usually be accomplished satisfactorily by using a perforated iron pipe placed in the flue at the desired point, but that for a sample of the flue gases to be representative it should be collected from a number of points in a given cross-section. The composition of the flue gas at any instant does not vary much at different points in a given cross-section; therefore the easiest method of sampling consists in using a tube provided with 2-mm. perforations so shaped that it will best distribute the points from which the gas is to be drawn for analysis.

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[1] Bull. No. 12 Bureau of Mines, J. C. W. FRAZER and E. J. HOFFMAN.

The usual method of sampling gas by means of a water bottle does not always lead to accurate analytical results, and this is specially true if the gas contains a considerable amount of carbon dioxide; due to the ease with which water dissolves this substance, there is a tendency to equalize the carbon dioxide in the samples collected, as illustrated by the following:

If water is saturated with a furnace gas containing 12 per cent. of carbon dioxide, and if this water is again used in collecting a sample of gas having a carbon dioxide content of 8 per cent., the water saturated with gas containing the 12 per cent. of carbon dioxide will give up some of the dissolved gas, and the consequent sample will be found to contain more than 8 per cent. of carbon dioxide. The magnitude of the error introduced by this method of sampling will vary with conditions, but it will depend in large measure upon the length of the sampling period and the total time of contact between the sample and the water.

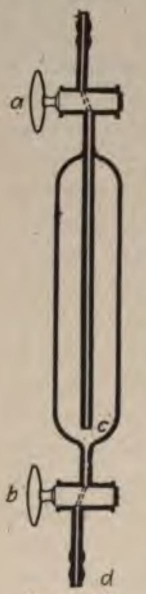

Fig. 67.—Gas sampling device.

Frazer and **Hoffman** recommend the following apparatus for collecting and storing a sample, and it has proved satisfactory in practical operations. The glass vessel, Fig. 67, is not only an important part of the sampling apparatus, but it also serves to store the sample after it has been collected. The device is made with a capacity of from 150 to 250 cc. If, with the vessel in the position shown in Fig. 67, it should be filled with mercury, and if the stopcocks (*a*) and (*b*) are opened, the mercury will flow from the lower tube, and the gas will be drawn in through the upper tube, entering the device at (*c*), and collecting above the mercury. So long as the surface of the mercury remains above (*c*) the same volume of gas will be collected in each equal interval of the sampling period, due to which, the sample collected will always be representative. The time required to drain a certain amount of mercury from the device can be varied from that when cocks (*a*) and (*b*) are completely open to a period of from 8 to 10 hours, or longer if desired, by attaching a short length of glass tubing, the outlet end being drawn out to a smaller diameter, to the tube at (*d*) by means of a rubber connection; by proper trial these short glass tubes can be so proportioned, as to their diameters, that the speed of the outflowing mercury can be regulated to any desired rate.

After the desired gas sample has been collected, the tube above (*a*) is filled with mercury by means of a funnel the stem of which has been drawn out to a capillary. The device is now inverted and, by means of a rubber tube attached to a reservoir of mercury, the inclosed gas is placed under a pressure of about 100 mm. of mercury. When the vessel is returned to the position shown in Fig. 67, the cocks (*a*) and (*b*) are mercury sealed, after which there is no possibility of the device leaking gas. In order to remove the sample for analysis, the vessel is again inverted and the tube above (*b*) is filled with mercury and attached to the burette of an analyzing instrument. By means of the same mercury reservoir which was used to produce a pressure on the gas, the desired volume of flue gas is forced out of the device and into the burette. If the gas is to be measured over water the tube above the stopcock (*b*) is filled with water instead of mercury.

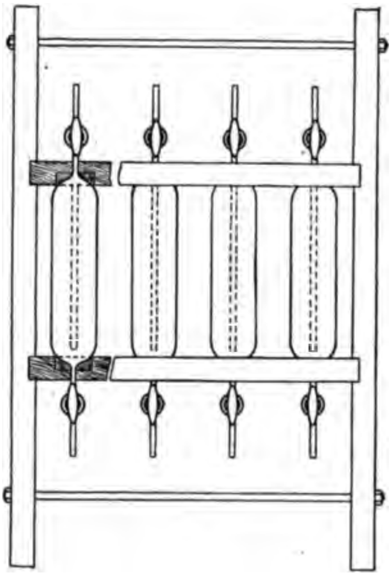

Fig. 68.—Stand for sampling tubes.

Frazer and **Hoffman** state that in order to facilitate the handling of these vessels, they should be mounted in a portable stand and in some cases it is desirable to arrange them in batteries of two to four each, Fig. 68 showing a convenient stand for mounting four of these tubes. When filled with mercury the tubes should be carefully supported on both ends, and the stand should be so constructed as to permit of removing any tube without disturbing the remaining ones.

FRAZER and HOFFMAN recommend that such portion of the sampling apparatus as is introduced into the furnace should be either a water-cooled metal tube or, better, a water-cooled quartz tube, the water-cooled metal tube found most satisfactory being shown at *A*, Fig. 69. The inner tube, through which the gas is collected, is kept cool by cold water which circulates through the surrounding tube and returns through the outside annular compartment. The only difference in construction when an inner tube of quartz is used is the use of asbestos packing to maintain a water-tight joint at each end. The quartz tube apparatus is more fragile than the metal tube, but it is to be preferred to the latter owing to its greater range of utility, and it may be placed in the fuel bed if desired. If the inner tube should have a sufficiently small diameter, the end projecting from the furnace may be attached directly to the vessel receiving the gas sample, but it is not always practicable to have the tube diameter so small. The end of the tube should be connected with a ¼-in. lead pipe, the latter being of a length which will reach to a table placed as close to the furnace as possible, and by connect-

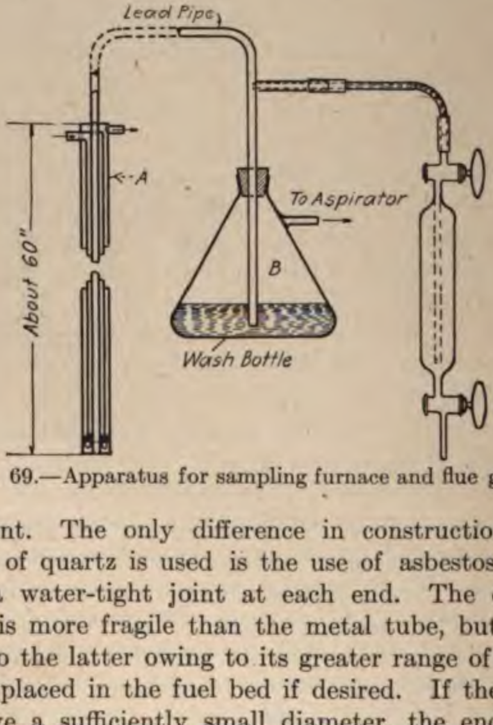

FIG. 69.—Apparatus for sampling furnace and flue gases.

ing the other end of the lead pipe to an aspirator, it becomes possible to draw a continuous current of gas from the furnace or flue. A rather rapid aspiration will ensure a sample whose composition will closely approximate that of the furnace, or flue, gas at the point reached by the open end of the tube. The lead pipe at the table end is perforated for the attachment of a small-bore glass tube, the latter being then connected directly with the mercury-filled sample receiver described above.

If the water-cooled tube should be inserted in the flame region of the furnace, it is found that the tube becomes clogged at times by the condensed tarry matter and by the molten ash which is carried into the tube by the rapidly moving gases, but the diameter of the tube should be large enough to prevent this. Again, it is possible that water vapor derived from the moisture in the fuel and from combustion of the hydrogen and its compounds, may also condense in the tube, and therefore the latter should be of such cross-section that the tendency to form a water column by capillarity may be easily overcome by the current of gas; if this should not be so, the water column may be formed and the gas flow be entirely stopped. The gas tube should have an internal diameter of $3/16$ in., and a trap should be introduced at a point beyond that at which the sample is taken from the lead pipe, in order to ensure the proper flow of gas through the pipe; this trap may be in the form of the wash bottle at *B*, Fig. 69.

A sample taken in the manner just described will represent only the average condition existing during the collection period at a certain space near the end of the sampling tube; therefore, in order to ascertain conditions existing throughout the furnace, or flue, it is necessary to multiply and to properly distribute the number of points from which samples are taken. For this purpose as many sampling tubes as are desired may be employed, locating them at different points, and collecting the separate samples simultaneously in their respective receivers. Only one aspirator is necessary for this purpose, and all of the lead pipes attached to the various sampling tubes are led to the table on which the sample receivers are placed.

It is necessary at times to secure a definite gas sample at a certain point in the furnace at a certain definite instant, because this sample permits of securing information regarding furnace conditions which might not be obtained with the sampling device explained above, and **Frazer** and **Hoffman** recommend the device shown in Fig. 70 for this purpose.

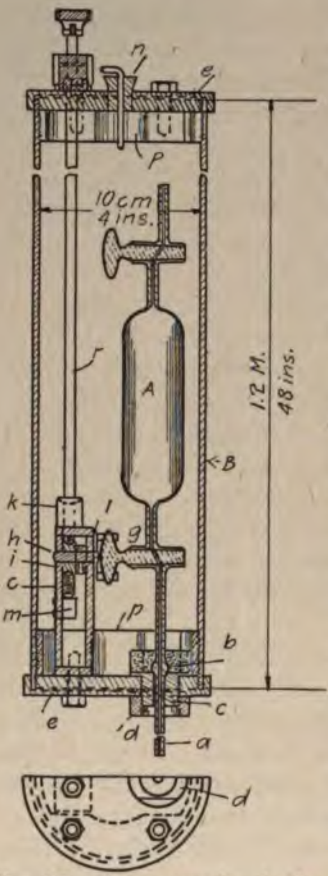

Fig. 70.—Apparatus for obtaining instantaneous gas samples.

This device consists of a quartz sampling tube (A) having a capacity of 100 cc., immersed in water contained in the steel tube (B), the latter having a diameter of 10 cm. and a length of 1.2 meters; at each end this vessel terminates in a thick-walled quartz tube, of 1 mm. bore, provided with a stopcock. One of these tubes (a) extends for a distance of 150 mm. beyond the stopcock, the open end projecting beyond the end of (B). An enlargement (b), located 60 mm. from the stopcock, provides a firmer hold for the cement of litharge and glycerin with which the cavity in the collar (c) is filled, (c) being thus permanently attached to (a). The opening and closing device for the sampling tube is made of brass and is located on the outside, supported by the two end pieces (e) of the steel tube (B). This opening and closing device consists of the brass frame (C), which supports the mechanism for turning the stopcock, this mechanism being made up of the brass shaft (h) carrying the wheel (i), and, beneath the frame, the brass plate (l) carrying four projections (g) which fit around the handle of the stopcock. In order to avoid straining the stopcock while turning, and which might occur if (h) were not centered directly above the stopcock, the projections (g) are fashioned in the shape of small rollers; the face of the wheel (i) is threaded to engage with the threaded end of the brass rod (r), while the piece (k) serves as a guide for the brass rod and provides means for adjusting the threaded end of the rod to the face of (i). The adjustment is accomplished by having the hole in (*k*), through which the rod (*r*) passes, eccentric to the bearing of (*k*) in (*C*); a movable stop (*m*) can be so set as to limit the rotation of (*i*) and the extent to which the stopcock may be turned. The steel tube (*B*) is threaded at each end on the inside for the purpose of receiving the threaded steel pieces (*P*), the latter being tapped for bolts which secure the end pieces (*e*). All joints leading to the interior of (*B*) are made tight by means of lead washers, on which the two surfaces are tightly drawn down by means of bolts.

Before collecting a sample with this device, the end piece (*e*), which is to carry the vessel (*A*), should be removed by removing the bolts which attach it to (*B*); the quartz tube is then placed in position and the nut is tightened. While this is being done (*c*) is prevented from turning by two small dowel pins which enter holes provided for them in (*e*). By trial (*m*) is so adjusted that when the rod (*r*) reaches it the handle of the stopcock is rotated through 90 degrees, and it is customary to adjust the stopcock so that communication with (*A*) is established when (*r*) is against (*m*), as shown in Fig. 70. The withdrawal of (*r*) then closes the stopcock. After the stopcock has been properly adjusted, the end piece (*e*) is bolted to (*B*), and the latter is suspended by means of a handle clamped on the other end, this handle not being shown in the illustration. While in a vertical position the rod (*r*) is introduced through the brass bushing at the upper end, and it is inserted in the hole in (*k*). The steel tube (*B*) is now filled with water through (*n*), the latter being then closed by means of a perforated rubber stopper, and through this perforation is passed a short glass tube bent at a right angle and having its projecting end directing upward when the apparatus is in use, this tube being for the purpose of relieving the internal pressure when the water temperature rises.

After being thus prepared the device is taken to a vacuum pump and placed in a horizontal position in two semicircular supports, the latter preventing its moving while connected to the pump. The tube (*A*) is now exhausted, and, while it is still connected to the pump, the rod (*r*) is withdrawn to a degree which will cause the threaded end to leave (*i*) but so that it will still remain in (*k*), thus closing the stopcock. The pump is now disconnected, after which the device should be introduced into the furnace as soon as convenient, the open end of the tube (*a*) being placed at the point from which the sample is to be taken.

The stopcock is opened by pushing in the rod (*r*) until it is in contact with (*m*), and after the short time required to fill the vessel (*A*) the rod is withdrawn and the stopcock thus closed, this operation requiring about 30 seconds.

After the sample is secured, and after the device has been removed from the furnace, the water is drained from (*B*), the rod (*r*) is withdrawn, the nut (*d*) is started to insure its subsequent easy removal, and the endpiece (*e*) carrying (*A*), is removed. Now (*d*) is entirely removed, and after the two glass tubes have been filled up to the stopcocks with mercury, the sample in the vessel (*A*) is transferred to one of the sample vessels, Fig. 67.

The results of the analyses of instantaneous samples give the conditions which temporarily prevail at the points where collected, but if it should be desired to co-ordinate the results of analyses with the conditions which prevail during the operation of the furnace, the continuous sampling method must be adopted. With the use of the "instantaneous" device shown in Fig. 70 it is easy to determine the amount of water vapor accompanying a gas sample, this being done by absorbing and weighing the moisture. If there should be any traces of nitric oxide present in the gas, their presence can readily be detected by a simple modification of the method described above, or as follows:

The sampling tube used consists of a water-cooled quartz tube of a construction similar to that shown in Fig. 69, an 8-liter evacuated bottle being used in place of the quartz vessel (*A*), Fig. 70, for receiving the gas sample. Two glass tubes with stopcocks are passed through each of two holes provided in the rubber stopper of the bottle, one of these tubes extending almost to the bottom of the bottle, the other extends only to immediately below the stopper, being bent at a right angle and connected directly with the water-cooled quartz tube. By following this arrangement the whole or a portion of the gas sample can be easily removed from the bottle, or any desired liquid can be introduced into the bottle without exposing the gas contained therein. Care should be taken to entirely cover the stopper with sealing wax in order to make the bottom gas-tight. After the sampling tube has been inserted into the furnace, or flue, the evacuated bottle is filled by opening the stopcock.

After the sample has been collected, an excess of an alkaline solution of potassium permanganate ($KMnO_4$) is introduced into the bottle and permitted to stand for 24 hours, at the end of which time the solution is drawn off and the free ammonia is distilled off. Potassium hydroxide (KOH) and fine aluminum powder are now added and this mixture is permitted to stand for several hours, after which the ammonia formed is distilled into standard sulphuric acid. If there should be any nitric oxide present, it can readily be determined qualitatively by introducing a solution of starch and potassium iodide into the bottle immediately after the sample has been collected, and if the resultant blue color should not appear immediately, due to the presence of sulphur dioxide in the gas, the color will become noticeable in a short time and will soon become quite pronounced. The presence of nitric oxide in the flue gas is important, because its formation is accompanied by the absorption of heat.

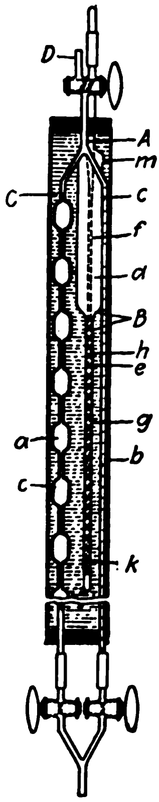

Fig. 71.—Burette and compensator.

Frazer and **Hoffman** recommend the burette shown in Fig. 71 for accurate analyses. This apparatus consists of the burette (*A*) and the automatic compensating device (*B*), the measuring portion of the burette and the entire compensator being enclosed in the water jacket (*C*). The measuring portion of the burette consists of the two limbs (*a*) and (*b*), the graduated portions of which, 66 cm. long, are united at the top in an inverted Y-shaped connection, a **Greiner-Friedrich** two-way stopcock being attached to the latter. By means of this stopcock communication can be established with either of two short, thick-walled tubes of small bore, one of which is connected with the compensating device. A second Y-tube is placed at the lower end of the device outside the water jacket (C), each of the limbs of this Y-tube being provided with a stopcock and attached by means of rubber connections to the projecting ends of the limbs (*a*) and (*b*). The lower end of this Y-tube is connected with a mercury reservoir by means of heavy rubber tubing. The tube (*a*) is made up of a series of 10 bulbs, each bulb having a capacity of 10 cc. between the two graduation marks located immediately above and below it, while the straight glass tube (*b*) has an internal diameter of about 4.5 mm., its graduated part having a total capacity of 10.1 cc., the beginning of the graduated portion of each limb of the burette being at (*c*). In the compensator (*B*) the confined air, the pressure of which at a constant volume is equalized by that of the gas to be measured, is contained in the bulb (*d*) and above the mercury surface (*e*) in the tube (*g*), the latter forming the lower termination of the bulb. The glass tube (*f*) is connected at its upper end with one of the passages through the stopcock of the burette, being sealed into the top of the bulb (*d*) while its other end extends nearly to the bottom of (*g*) and opens beneath the surface of the mercury. This tube is centered in (*g*) at two points (*h*) and (*k*), the former located 25 mm. above the surface of the mercury, and the latter about 50 mm. from its lower end, this centering consisting of three small glass projections sealed on the outside of the tube at each of these points. The diameters of the tubes (*f*) and (*g*) are so proportioned that the distance from the inside of (*g*) and the outside of (*f*) is as nearly as possible equal to the internal diameter of (*f*). The distance from the surface of the mercury in (*f*) to the stopcock of the burette should be as short as possible, because the gas in this portion of the tube must be drawn into the burette by a decrease in pressure. The depth of mercury in the compensator should be sufficient to require about one-third of an atmosphere excess pressure in the burette to force the gas from the latter into the compensator, but care should be taken not to have the quantity of mercury so great that its expansion or contraction will produce an appreciable change in the volume of the inclosed air. The tube (*f*) should extend nearly to the bottom of the compensator in order to displace as much mercury as possible.

The compensator may be filled in such manner that the readings on the burette show the correct volumes of the gas at 0°C. and 760 mm. pressure, or it may be closed under known conditions of temperature and pressure and the consequent readings corrected to standard conditions. The latter method is the one adopted for ordinary work, and in this case the compensator may be closed by replacing the seal at (*m*) by a small tube and stopcock. The limb (*b*) of the burette, and on which the readings are made, is graduated in hundredths of 1 cc., and tenths of these divisions can be estimated quite accurately.

Before taking a reading of a gas volume in the burette, it is necessary to stir the water in the jacket (*C*) and to allow a few minutes to elapse in order to be sure that the gas is at the temperature of the surrounding water. The gas is now nearly all drawn into the limb (*a*), and the pressure is adjusted to approximately that of the atmosphere, by which means the number of bulbs the gas will fill completely, when at the pressure of the air in the compensator, is ascertained. After this has been determined, the mercury in the burette is brought to the exact level of the graduation beneath the last bulb completely filled in the trial experiment, then closing the stopcock at the bottom of the limb. The remaining fraction of a bulb full of gas is then caused to enter the limb (*b*), and the two-way cock is turned so as to establish communication between the burette and the compensator. The pressure of the gas in the burette should now be so adjusted, by means of the mercury reservoir, that the two surfaces of mercury in the compensator are on the same level, the pressure of the gas in the burette being then equal to that of the gas in the compensator. The stopcock at the bottom of (*b*) is now closed and the reading of the burette is taken. To this reading must be added the predetermined capacity (*K*) of that portion of the apparatus between the graduated portion of each limb of the burette and the mercury meniscus in the tube (*f*). As each constituent of the gas is determined by the difference in the burette readings, before and after an absorption, this constant capacity (*K*) does not enter into the determination of the amount absorbed, but it is necessary to apply this correction in order to obtain the initial volume of the gas unless exactly sufficient nitrogen to fill this portion of the apparatus is taken into the burette previous to the introduction of the gas sample.

Frazer and Hoffman state that the reading of this burette requires some practice to secure the best results. The gases are measured in their moist condition, and the quantity of water introduced into the burette and compensator to effect this must be only sufficient to moisten the glass walls of the tubes. If there should be sufficient water in the compensator to drain down upon the mercury, the accurate adjustment of the mercury surface is rendered difficult; too much water in the burette tends to stop up the narrow tubes forming the connections between the bulbs, and thus interferes seriously with the distribution of pressure on the gas in the burette and consequently with the equalization of the pressures in both burette and compensator.

A complete gas analysis can be made with this apparatus by connecting the burette in turn with the various absorption pipettes, but it is more convenient and accurate to connect the burette permanently with the pipettes, errors due to the repeated making and breaking of connections being thus avoided.

The entire system may be mounted in a wooden stand, and a series of three pipettes can be connected with the tube (*D*) by means of fine capillary tubing in a manner similar to that practised with the ORSAT apparatus. These pipettes consist of a combustion pipette provided with an electrically heated platinum coil, an absorption pipette containing a solution of potassium hydroxide for the absorption of carbon dioxide, and a pipette containing phosphorus for the oxygen determination. In order to avoid any errors due to the capacity of the connecting tubes, they should be made as short as possible, and their internal diameter should not exceed 0.4 mm., but all error from this source can be avoided by an actual determination of the capacity of these connections, or by filling them and that portion of the apparatus between the ungraduated portions of the burette and the mercury meniscus in the tube (*f*) with pure nitrogen before introducing the gas sample into the burette.

BURRELL and SEIBERT[1] state that the *exact* analysis of the combustible constituents in flue gas has received little attention, this being largely due to the difficulty of readily determining the small quantities of these constituents usually present. This is especially true of hydrogen and methane, the usual flue gas analysis being made for carbon dioxide, carbon monoxide, and oxygen only. In order to remedy such defective analysis, they devised the apparatus shown in Fig. 72. This apparatus is a modification of the one used by HALDANE in investigating the air in mines, it being very much simplified from the original in that all pipettes are placed in a common train. The three-way stopcocks of HALDANE have been replaced by three two-way stopcocks, and a simpler pipette has been provided for the alkaline pyrogallate solution. The burette (*d*) contains mercury, its total capacity being 21 cc.; the bulb of the burette, which is not graduated, has a capacity of 15 cc., while the stem of the burette is graduated to 0.01 cc. and has a capacity of 6 cc. The three pipettes, Fig.

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[1] Tech. Paper 31, Bureau of Mines, G. A. BURRELL and F. M. SEIBERT.

72, are the potassium hydroxide (*a*); the slow combustion (*b*); and the alkaline pyrogallate (*c*); the reservoir of the alkaline pyrogallate pipette is located behind the stand and is provided with a rubber bag for the purpose of preventing access of air to the alkaline pyrogallate solution.

Care must be taken in all cases to see that the surfaces of the liquids in the reservoir bulbs are at the same level as the surfaces of the liquids in the corresponding pipettes when the liquids are brought to the marks shown at (*e*), (*f*) and (*g*), in order to prevent a difference in pressure acting against the columns of the liquids in the pipettes when the sample of gas is transferred from the burette to the different pipettes.

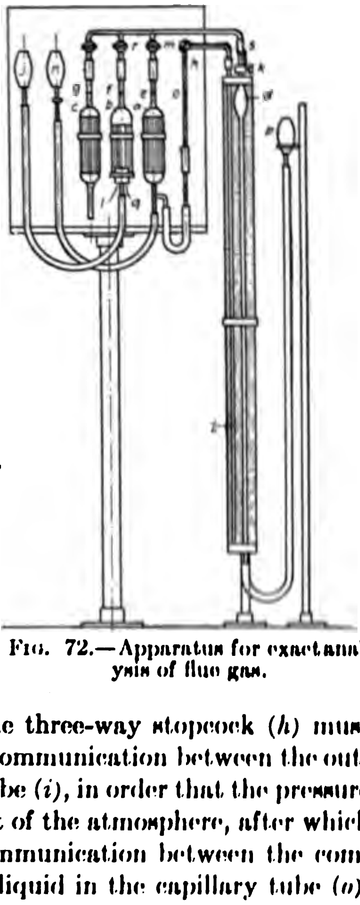

FIG. 72.—Apparatus for exact analysis of flue gas.

BURRELL and **SEIBERT** explain the operation of the apparatus as follows:

In order to perform the analysis, the liquids in the three pipettes must be brought exactly to the marks (*e*), (*f*) and (*g*) on the capillary tubes, but if a recent analysis has been made, this preliminary step will not be necessary; if the apparatus has been idle for some time this adjustment will have to be made before beginning the analysis. The three-way stopcock (*h*) must now be turned so as to establish communication between the outside air and the compensating tube (*i*), in order that the pressure in the tube may be equal to that of the atmosphere, after which it is again turned to establish communication between the compensator (*i*) and the column of liquid in the capillary tube (*o*). The gas sample is now transferred to the burette by displacing it with mercury, as explained before. The first 5 or 10 cc. of gas drawn into the burette (*d*) from the sample container are not retained, but are used to displace and thus remove the air from the connections between the burette and the sample container, an extra three-way cock being placed between the burette and the sample container for this purpose. Gas is now drawn into the burette until the mercury is depressed somewhat below the 21-cc. mark, after which the burette stopcock (*k*) is closed, and the gas is placed under pressure by slightly raising the level bulb (*p*). The burette stopcock (*k*) is then opened to the air for a second, thus bringing the gas in the burette to atmospheric pressure, and before reading the volume of gas inclosed, the stopcocks (*k*) and (*m*) must be turned so as to establish communication between the burette and the potassium hydroxide solution pipette (*a*).

If these directions have been carefully followed, only a slight movement of the potassium hydroxide solution at (*e*) will be seen, but if proper care should not be taken, a decided movement of the solution will occur, with a consequent error. A resulting increase of pressure causes an absorption of some carbon dioxide before a reading can be taken, while a decrease in pressure will draw the potassium hydroxide solution into the horizontal capillary tubing, thus causing such error in the combustion data that its value will be of no account. A slight raising or lowering of the level bulb (*n*) will bring the surface of the potassium hydroxide solution exactly to the mark (*o*) in the capillary tube.

A slight movement of the mercury in the burette will bring the potassium hydroxide solution to the mark (*e*) above the pipette, and the volume of gas can be read to the third decimal place; the burette can be read to 0.002 cc. The gas is not passed into the potassium hydroxide pipette, and the carbon dioxide will be entirely removed by passing the gas back and forth between the burette and the potassium hydroxide pipette about four times. Under no condition should the mercury in the burette be raised above the stopcock (*k*). The reduction in volume of the gas sample should now be recorded, after which the sample is passed into the slow combustion pipette (*b*). The platinum coil in this pipette is of No. 30 B. and S. wire, and it is in contact with platinum wires in two mercury-filled glass tubes, each wire being sealed through the bottom of its tube, the projecting ends being attached to copper wires for carrying the electric current, the coil being thus brought to a white heat by a current of about 5 amp. Two minutes are required for the complete oxidation of the combustible gases contained in the sample. The gas remaining in the capillary tubing is brought into contact with the platinum spiral by passing the gas back and forth between the burette and the combustion pipette several times, but care must be taken to see that the mercury in the combustion pipette is not raised to the platinum spiral in the pipette. The current is now broken and the pipette is allowed to cool, cooling being hastened by playing a stream of compressed air upon the pipette; after the pipette has been cooled, the gas is transferred to the burette and the decreased volume is recorded, the residual gas being then passed into the potassium hydroxide pipette (*a*) for the absorption of any carbon dioxide produced by the combustion. After the decrease in volume caused by this absorption has been measured, the gas is passed into the pipette (*c*) containing the alkaline pyrogallate solution in order to absorb the oxygen.

It may be assumed that in this stage of the analysis the capillary tubes will be filled with residual gases containing oxygen; therefore, in order to avoid sweeping them out by dilution, and in order to obtain the last trace of oxygen, the mercury in the combustion pipette must be brought to the stopcock (*r*) by raising the bulb (*j*), while the potassium hydroxide solution is brought to the stopcock (*m*) by raising the bulb (*n*). The oxygen can be completely absorbed by passing the gas back and forth between the burette and the alkaline pyrogallate solution about six times. Finally, the alkaline pyrogallate solution must be brought exactly to the mark (*g*), while the mercury and the potassium hydroxide solution are brought to their respective marks at (*f*) and (*e*), the reduction in gas volume being then measured. The oxygen percentage determined in this manner will represent the oxygen remaining after a definite volume has been consumed in the burning of the combustible constituents. A separate determination is necessary to determine the volume of oxygen consumed; this is done by drawing a fresh sample of gas into the burette and then passing it in turn into the potassium hydroxide and alkaline pyrogallate solutions, and then measuring the reduction in volume thus produced. The total oxygen content thus determined, minus the residual oxygen previously found, will represent the oxygen consumed.

Other burette readings are made in a manner similar to that in which the original volume of the sample was read; for instance, after combustion the gas is withdrawn to the burette and the mercury in the combustion pipette is brought exactly to the mark at (*f*), the stopcock at (*r*) being then closed and the one at (*m*) opened. The solution in the capillary tube is then brought to the mark at (*o*) by raising or lowering the level bulb (*n*) and the potassium hydroxide solution in (*a*) is brought to the mark (*e*) by a slight movement of the mercury in the burette, after which the burette is read. From the contraction, the carbon dioxide produced by the combustion and the oxygen consumed, the methane, carbon monoxide, and hydrogen may be calculated from the following equations:

$$\left.\begin{array}{l} CH_4 + 2O_2 = CO_2 + 2H_2O \\ 2H_2 + O_2 = 2H_2O \\ 2CO + O_2 = 2CO_2 \end{array}\right\} \qquad (50)$$

CHAPTER IX

PYROMETRY

In order to make proper furnace tests it is necessary that the temperatures in various portions of the furnace should be measured, because there is a wide range in temperature between the furnace, the setting, and the waste gas flue. The most important methods for determining these temperatures are:

1. The mercurial pyrometer, which is capable of measuring temperatures up to 1000°F. (540°C.), is generally used for taking flue-gas temperatures, owing to its accuracy and ease of manipulation. Due to the fact that at atmospheric pressure mercury will boil at 676°F. (358°C.), and as even at lower temperatures the mercury in the thermometer will be distilled and will thus collect in the upper portion of the stem, it is necessary that for temperatures exceeding 400°F. (205°C.) an inert gas, such as nitrogen or carbon dioxide, be forced under pressure into the upper portion of the stem. At 600°F. (315°C.) the pressure amounts to about 15 lb., or a little above that of the atmosphere, while at 850°F. (455°C.) this pressure will be about 70 lb., and at 1000°F. (540°C.) it will be about 300 lb.

The use of this pyrometer requires, however, that great care be taken to see that the mercury bulb is inserted in the direct path of the moving gases, otherwise the temperature readings will not be true records of the flue-gas temperature, and no readings should be accepted until the thermometer has been in position for a period sufficient to heat it up to the full-gas temperature.

2. Expansion pyrometers are used for recording temperatures up to 1500°F. (815°C.). The principle of this pyrometer is based on the difference in expansion of two metals, brass and iron being generally used for this purpose. Brass expands about 50 per cent. more than iron through the same range of temperature, and the expansion of both brass and iron is practically proportional to the increase in temperature. This pyrometer is usually made by enclosing a brass rod in an iron pipe, one end of the rod being secured in a rigid manner to a cap on the end of the iron pipe, the

ing connected by a multiplying gear to a
to move around a graduated dial.
ry reliable: first, because it is necessary
the expansion piece be at a uniform tem-
urate reading can be secured; and second, be-
some lost motion in the mechanism connect-
ometer. These conditions require that before
ometer should be carefully calibrated and its
e thoroughly understood. Unless the brass
pe are of the same temperature, unreliable
sult. If the instrument is allowed to cool after
sed to a high temperature, the needle will rise be-
o fail, and conversely, a rise in temperature will
as a decrease in temperature, these anomalous
ing due to the fact that the iron pipe, which is lo-
outside, will heat or cool in less time than the brass

[illegible] is used for temperatures up to 2000°F. (1095° C.),
its name from the fact that the process used is simi-
[illegible] of the specific heat of a substance by the
[illegible], the only difference being that in one case the
[illegible] known and the specific heat is sought, while in the
the specific heat is known and the temperature is

[illegible] determine temperatures by this method, a known
some substance such as iron, nickel, or firebrick must be
e unknown temperature and then plunged into water,
temperature of the water being recorded.

[illegible] - the temperature to be measured,
- the weight of the heated body in pounds,
- the weight of the water in pounds,
- the final temperature of the water,
- the difference between the initial and the final temperatures of the water,
- the known specific heat of the body,

[illegible]

$$T = \frac{T' + Wt}{wS} \tag{51}$$

[illegible] this method of temperature determination is only
[illegible] because errors are prone to result from many causes
[illegible] the command of the operator.

4. Thermo-electric pyrometers are used for temperatures up to 2900°F. (1595°C.). This instrument is based on the phenomena produced when two wires of different metals are joined at one end and heated, this conjunction setting up an electromotive force between the free ends of the wires. The amount of the electromotive force produced will depend upon the composition of the wires and upon the difference in temperature between the two. This combination between the two wires is termed a "thermo-couple." If a very delicate galvanometer of high resistance is connected to this thermo-couple, the current produced will cause a deflection of the needle, and if the graduated dial under the needle is properly calibrated, an accurate temperature reading will result.

There are quite a number of thermo-electric pyrometers on the market, but that of Le Chatelier is probably the one most used. In this instrument the wires of the couple are of platinum and a 10 per cent. alloy of platinum and rhodium, these wires being enclosed in porcelain tubes in order to protect them from the oxidizing influence of the furnace gases, this protected couple being termed an "element."

Accuracy does not require that the entire length of the element be exposed to the temperature to be measured, because the electromotive force produced depends only upon the temperature of the juncture at the closed end of the protecting tube and upon the temperature of the cold end of the "element." This instrument is very accurate over a wide range of temperatures, and as

Table LVII.—Approximate Melting Points of Metals

Metal	Temperature		Metal	Temperature	
	°F.	°C.		°F.	°C.
Wrought iron	2737	1500	Lead	621	325
Gray pig iron	2190 2327	1200 1255	Bismuth	498	259
White cast iron	2075	1135	Tin	449	232
Steel	2460 2550	1350 1400	Platinum	3191	1755
Cast steel	2500	1370	Gold	1946	1060
Copper	1981	1085	Silver	1762	960
Zinc	786	420	Aluminum	1216	660
Antimony	1166	630			

the galvanometer can be located at almost any convenient point, continuous readings can be easily secured, but it must be remembered that the length of the wires leading to the galvanometer alter the resistance of the circuit, and consequently proper allowance must be made for this condition.

5. The melting points of various metals which flow at various temperatures up to the melting point of platinum, 3191°F. (1755°C.), are also used for high temperature determinations. The approximate melting points for the various metals are given in Table LVII.

TABLE LVII-A.—MELTING POINTS OF SEGAR CONES

Cone number	Melting point		Cone number	Melting point	
	°F.	°C.		°F.	°C.
022	1,094	590	9	2,390	1,310
021	1,148	620	10	2,426	1,330
020	1,202	650	11	2,462	1,350
019	1,256	680	12	2,498	1,370
018	1,310	710	13	2,534	1,390
017	1,364	740	14	2,570	1,410
016	1,419	773	15	2,606	1,430
015	1,472	800	16	2,642	1,450
014	1,526	830	17	2,678	1,470
013	1,580	860	18	2,714	1,490
012	1,634	890	19	2,750	1,510
011	1,688	920	20	2,786	1,530
010	1,742	950	21	2,822	1,550
09	1,778	970	22	2,858	1,570
08	1,814	990	23	2,894	1,590
07	1,850	1,010	24	2,930	1,610
06	1,886	1,030	25	2,966	1,630
05	1,922	1,050	26	3,002	1,650
04	1,958	1,070	27	3,038	1,670
03	1,994	1,090	28	3,074	1,690
02	2,030	1,110	29	3,110	1,710
01	2,066	1,130	30	3,146	1,730
1	2,102	1,150	31	3,182	1,750
2	2,138	1,170	32	3,218	1,770
3	2,174	1,190	33	3,254	1,790
4	2,210	1,210	34	3,290	1,810
5	2,246	1,230	35	3,326	1,830
6	2,282	1,250	36	3,362	1,850
7	2,318	1,270	37	3,398	1,870
8	2,354	1,290	38	3,434	1,890

The use of this method requires the introduction of a metal, the melting point of which is known, into the furnace, and by using a series of metals whose melting points vary from 100°F. to 200°F., the temperature can be determined as existing somewhere between some two of these metals, but this method is very inaccurate and serves only for an approximation. Segar cones made of refractory material of various constituency, and whose exact melting points are known, are also used for this purpose. Segar cones are moulded into triangular pyramids, the various melting temperatures being secured by varying the mixtures employed in the manufacture of the cones; Table LVII-A gives the melting points of the cones used in pyrometry.

6. Radiation pyrometers are used for determining temperatures up to 3600°F. (1995°C.). This instrument is very similar to the thermo-electric pyrometer, and it also makes use of the thermo-couple. The heat rays projected from the hot body, whose temperature is to be measured, fall on a concave mirror and are then brought to a focus at a point where the junction of a thermo-couple is located, the temperature readings being obtained by means of an indicator similar to that supplied with the thermo-electric pyrometer.

7. Optical pyrometers are used for measuring very high temperatures, and the one designed by WANNER is probably the most reliable. This instrument is constructed on the principle of comparing the quantity of light emanating from the body whose temperature is to be measured, with a constant source of light, the WANNER instrument making use of a 2-volt osmium lamp for this purpose. The instrument is provided with an optical tube, the lamp being placed at one end of the tube, the other end being provided with an eyepiece and a scale, the current for the lamp being supplied by a battery of cells. A circle of red light appears upon looking through the pyrometer, this circle being divided into distinct halves of different intensities. The instrument must be so adjusted that the two halves of the circle blend and appear exactly alike, when a reading can be taken from the scale, the temperatures being then obtained from a table corresponding to the scale readings. The instrument is usually provided with an amylacetate lamp for standardizing the osmium lamp.

WHITE and TAYLOR[1] compiled the data given in Table LVIII, which shows the character of emitted light and its corresponding approximate temperature; the color given off at a stated temperature is approximately the same for all kinds of combustibles under similar conditions.

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[1] Trans. A. S. M. E., 1900.

TABLE LVIII.—TEMPERATURE OF EMITTED LIGHT

Character of emitted light	Temperature		Character of emitted light	Temperature	
	°F.	°C.		°F.	°C.
Dark red, blood red, low red	1050	565	Light orange	1725	940
Dark cherry red	1175	635	Yellow	1825	995
Cherry, full red	1375	745	Light yellow	1975	1080
Light cherry, bright cherry, light red	1550	845	White	2200	1205
Orange	1650	900			

CHAPTER X

EUCHÉNE'S "THERMIC REACTIONS"[1]

M. Euchéne presented this most valuable data on the Thermic Reactions which occur during the distillation of coal before the International Gas Congress in Paris in 1900, and as his data are so valuable to a proper understanding of this most interesting problem, it is deemed advisable to again present the principal portions of his communication in these pages, all data being based on a heat balance which shows the heat generated on one side and the heat expended on the other, the difference found between these two quantities therefore representing the heat of decomposition of the coal.

For every 100 parts by weight of coal carbonized, a certain quantity of fuel must be consumed, this quantity of fuel developing a certain number of heat units which must be accounted for. We will let the heat developed by the fuel in question be represented by A calories per 100 parts by weight of the coal carbonized, and we will trace some of these calories: (1) As heat carried away by the volatile matter, *e.g.*, gas, water, tar, etc., representing this portion of the heat by D. (2) As heat carried away by the waste gases in the flue $= C$. (3) As heat retained by the coke $= E$. (4) As heat lost by radiation $= F$. (5) As heat carried away by the ashes and clinker $= H$. Under these conditions we find that the difference $A - (C + D + E + F + H)$ will represent the heat required for the decomposition of the coal, and we will represent this quantity of heat by x. Our equation will then read

$$x = A - (C + D + E + F + H) \qquad (52)$$

This reaction represents evident facts, but an analysis of the operation of coal distillation shows that it involves reactions which afford a very interesting study, and these internal reactions must be considered in the light of thermo-chemical data.

As stated in the preceding pages, coal consists of a mass of

[1] Adapted from Jnl. of Gas Lighting, London, 1900.

ternary compounds containing carbon, hydrogen, and oxygen as the principal constituents, with a little nitrogen and sulphur, and it is ordinarily associated with some water and earthy matter. All of these constituents, with the exception of moisture, are in a solid state, and they undergo transformation into solid and volatile products when the coal is distilled. The solid products consist of carbon, sulphur, and ash, with traces of nitrogen, hydrogen, and oxygen, while the volatile products comprise carbon dioxide, carbon monoxide, marsh gas, hydrogen, benzene, as well as ethylene and its homologues, these bodies forming the chief constituents of the gas; hydrogen sulphide, carbon bisulphide, hydrocyanic acid, thiocyanic acid, and hydrochloric acid occur only in small quantities, and they are usually extracted from the gas by external means; water, with some ammonia; and tar consisting of various compounds of carbon, of which naphthalene is conspicuous. While in the coal these constituents were in a solid state, but distillation, in the first place, decomposes the ternary compounds and volatilizes a number of the bodies. A portion of the carbon is converted into carbon dioxide and carbon monoxide by combining with oxygen, and into marsh gas, benzene, ethylene, and tar by combining with hydrogen. A portion of the hydrogen also combines with oxygen to form water, and other minor changes produce ammonia, hydrogen sulphide, etc.

The formation of these bodies is attended by the liberation or the absorption of heat, and it therefore affects the heat produced or the heat expended.

Table LIX gives the heats of formation of the chief constituents of gas.

TABLE LIX.—I. SUBSTANCES FORMED WITH LIBERATION OF HEAT

Substance formed	Liberates		Substance liberated
	Calories per kg.	B.t.u. per pound	
Carbon dioxide, CO_2	8,080	14,692	Amorphous carbon
Carbon monoxide, CO	2,473	4,496	Amorphous carbon
Marsh gas, CH_4	1,791	3,256	Amorphous carbon
	or 1,343	or 2,442	Marsh gas
Water (liquid), H_2O	34,462	62,660	Hydrogen
Hydrogen sulphide, H_2S	135	246	Hydrogen sulphide
Ammonia, NH_3	717	1,304	Ammonia

II. Substances Formed with Absorption of Heat

	Absorbs		Substance absorbed
Benzene, C_6H_6	125	228	Amorphous carbon
	or 115	or 211	Benzene vapor
Ethylene, C_2H_4	517	940	Amorphous carbon
	or 443	or 804	Ethylene
Carbon disulphide, CS_2	411	760	Carbon disulphide
Cyanogen, CN	1,319	2,400	Cyanogen
Tar (mean composition)	300	544	Tar

Tar may be regarded, and that without great error, as having a mean heat of formation equal to that of naphthalene—*i.e.*, 300 calories.

As all coals do not act in the same manner, it is necessary to determine what coal is to be used in producing gas, or in making coke, before a heat balance can be established. Wood, peat, lignite, and gas coal, when distilled under similar conditions, yield gas and secondary products similar in character, but on close examination they show remarkable differences. The classification of fuels as established by Regnault was confirmed by Mahler, and Euchéne used a table, based on the work of these two investigators, which shows the elementary composition of typical fuels, and then prepared Table LX showing St. Claire Deville's classification of gas coals, the figures in the table referring to the substance of the coal on an ash- and moisture-free basis.

Table LX.—Composition of Gas Coals, Euchéne

Coal type	Volatile matter, per cent.	Elementary composition			
		Carbon	Hydrogen	Oxygen	Nitrogen
No. 1	29.48	88.38	5.06	5.56	1
No. 2	33.88	86.97	5.37	6.66	1
No. 3	36.42	85.89	5.40	7.71	1
No. 4	40.66	83.37	5.53	10.10	1
No. 5	43.99	81.66	5.64	11.70	1

Euchene states that the percentage of nitrogen present does not change progressively, but that it varies with conditions which are not material for the present purpose, and he has therefore taken it as being constant. The volatile matter increases from type No. 1 to type No. 5, and it is therefore to be expected that the quantities of heat liberated and absorbed through the formation of volatile should increase with the proportion of the latter, and **Euchène's** calculations confirm this anticipation. These coals, as employed in gas making, contain ash and moisture, and **Euchène** gives tables showing the products of distillation, with

TABLE LXI.—PRODUCTS OF DISTILLATION WITH HEAT ABSORBED AND LIBERATED

Type 3 Coal. Based on 100 Parts (by Weight) of the coal

Consist of	Forming on distillation				Parts
3.31 parts moisture.....		Ammoniacal liquor			3.310
7.21 parts ash		Ash			7.210
76.00 parts carbon	Solid	Coke			61.611
		Retort carbon			0.100
	Liquid	Tar			4.687
	Gaseous	Carbon dioxide	0.284	9.564	9.602
		Carbon monoxide	1.352		
		Marsh gas	5.601		
		Benzene	1.016		
		Ethylene	1.311		
		Cyanogen	0.036	0.038	
		Carb. bisulphide	0.002		
4.78 parts hydrogen	Solid	Coke			0.469
	Liquid	Ammoniacal liquor			0.388
		Tar			0.327
	Gaseous	Hydrogen	1.366	3.537	3.596
		Marsh gas	1.867		
		Benzene	0.086		
		Ethylene	0.218		
		Hyd. sulphide	0.019	0.059	
		Ammonia	0.040		
6.82 parts oxygen	Liquid	Tar			0.436
		Ammoniacal liquor			3.107
	Gaseous	Carbon dioxide	0.758		2.560
		Carbon monoxide	1.802		
		Other products (?)			0.717
0.88 part nitrogen	Solid	Coke			0.270
	Gaseous	Gas	0.385		0.610
		Am. liquor	0.183		
		Cyanogen	0.042		
1.00 part sulphur	Solid	Coke			0.683
	Gaseous	Hyd. sulphide	0.306		0.317
		Carb. bisulphide	0.011		
100.00 parts					100.000

the quantities of heat liberated and absorbed, for the five types of coal given above and for peat. Coal of type 3 was used for exhaustive experiments, and therefore the significant portions of these data are given in Table LXI for this coal.

Grouping these products in the usual manner, we find that the 100 parts (by weight) of the coal, yielded:

	Parts by weight	
Coke:		
Ash	7.210	
Carbon	61.611	
Hydrogen	0.469	
Sulphur	0.683	
Nitrogen	0.270	
		70.243
Gas:		
Carbon dioxide	1.042	
Carbon monoxide	3.154	
Hydrogen	1.366	
Marsh gas	7.468	
Nitrogen	0.385	
Benzene	1.102	
Ethylene	1.529	
		16.046
Ammoniacal Liquor:		
Moisture in coal	3.310	
Formed	3.495	
		6.805
Tar:		
Carbon	4.687	
Hydrogen	0.327	
Oxygen	0.436	
		5.450
Hydrogen sulphide		0.325
Ammonia		0.223
Retort carbon		0.100
Cyanogen		0.078
Carbon bisulphide		0.013
Other products, or unaccounted for		0.717
Total		100.000

The yield of gas amounted to 30.64 cubic meters per 100 kilos (4.909 cu. ft. per pound), or 10,995 cu. ft. per ton of 2240 lb., while the composition of the gas in volumes per cent. was:

Hydrogen	50.10
Marsh gas	34.03
Carbon monoxide	8.21
Ethylene	3.98
Carbon dioxide	1.72
Nitrogen	1.00
Benzene	0.96
	100.00

The heat liberated, as well as the heat absorbed in the formation of compounds per 100 kilos, or per 100 lb. of coal distilled, is given in Table LXII.

M. Euchêne gives similar data for the other four types of coal as well as for peat, the more important figures from his summaries being given in Table LXIII. A careful examination of these tables shows:

1. That there is a very close agreement between the results of the elementary analyses and the practical results; a slight excess of oxygen is found, and this is doubtless due to the fact that the oxygen was estimated by difference.

2. That the heat liberated during distillation ranged from 76,564 calories for peat to 23,718 calories for coal (139,208 to 43,124 B.t.u.) containing little oxygen; and that the heat absorbed varies from 4333 calories for peat to 1976 calories for coal (7860 to 3593 B.t.u.) containing little oxygen. The heat liberated is, therefore, greatly in excess of the heat absorbed.

These facts, however, do not permit of drawing any conclusion as to the amount of heat required for the distillation of the coal, because they give no idea of the value of the term which represents the heat of decomposition of the coal. This value is finally found by difference. One piece of information, however, is afforded by the practical results, viz., that the consumption of fuel varies in inverse proportion to the amount of heat liberated by the volatile products.

3. That the fixed residue of coke increases progressively from peat to the coal containing the least amount of oxygen; and that the nature of the residue changes from a non-caking wood charcoal (from peat) to a well-caked large coke (from the coal containing the least amount of oxygen). Also, that the rapidity of the distillation falls off from peat to the coal containing the least amount of oxygen.

TABLE LXII.—HEAT LIBERATED PER 100 KILOS, OR PER 100 LB. OF COAL DISTILLED

Compund	Amount		Containing	Kilos	Pounds	Heat of formation		Heat liberated		Heat absorbed	
	Kilos	Pounds				Calories	B.t.u.	Calories	B.t.u.	Calories	B.t.u.
Carb. dioxide	1.042	0.472	Carbon	0.284×	0.120×	8,080	32,320	2,295	3,878		
Carb. monoxide	3.154	1.800	Carbon	1.352×	0.614×	2,473	9,892	3,343	6,074		
Marsh gas	7.468	3.400		×	×	1,343	5,372	10,030	18,265		
Hyd. sulph.	0.325	0.148		×	×	135	540	44	80		
Ammonia	0.223	0.101		×	×	717	2,868	160	290		
Water	3.495	1.590	Hydrogen	0.388×	0.176×	34,462	137,848	13,371	24,261		
Total								29,243	52,848		
Benzene	1.102	0.500			×	115	460			127	230
Ethylene	1.529	0.700			×	433	1,732			677	1,212
Cyanogen	0.078	0.035			×	1,319	5,276			103	185
Carb. bisulph.	0.013	0.006			×	411	1,644			5	10
Tar	5.450	2.500			×	300	1,200			1,635	3,000
Total										2,547	4,637

Table LXIII.—Coal Products

Type of coal, etc.	Elementary composition							Volatile matter obtained on distillation. Percentage by weight			
	Moisture	Ash	Carbon	Hydrogen	Oxygen	Nitrogen	Sulphur	Gas	Tar	Water	Total
No. 1........	2.17	9.040	77.790	4.460	4.890	0.880	0.770	14.283	4.500	4.584	23.367
No. 2........	2.70	7.060	77.560	4.790	5.940	0.890	1.060	15.361	5.070	5.567	25.998
No. 3........	3.31	7.210	76.000	4.780	6.820	0.880	1.000	16.046	5.450	6.805	28.301
No. 4........	4.34	8.180	72.080	4.780	8.730	0.870	1.020	17.258	6.300	8.616	32.174
No. 5........	6.17	10.730	67.010	4.630	9.600	0.820	1.040	16.797	6.400	9.861	33.058
Peat.........	17.50	5.342	47.475	5.207	22.975	0.825	0.676	36.250	8.300	26.500	71.050

Table LXIII.—Coal Products.—*Continued*

Type of coal, etc.	Heat liberated during distillation				Heat absorbed during distillation				Coke from coal, per cent.	Carbon volatilised, per cent. of total carbon	Fuel coke consumed, per cent. of coal distilled	Nature of coke produced
	Calories per 100 kilos		B.t.u. per 100 lb.		Calories per 100 kilos		B.t.u. per 100 lb.					
	Coal as distilled	Coal less moisture and ash	Coal as distilled	Coal less moisture and ash	Coal as distilled	Coal less moisture and ash	Coal as distilled	Coal less moisture and ash				
No. 1......	23,718	26,712	43,124	48,696	1,976	2,225	3,593	4,046	75.661	15.50	....	Caked, dense, large
No. 2......	26,228	29,064	47,768	52,844	2,282	2,529	4,149	4,598	72.567	17.40	22.40	Caked, dense
No. 3......	29,243	32,681	53,176	59,420	2,547	2,846	4,631	5,174	70.243	18.80	21.00	Caked, medium size
No. 4......	34,213	39,109	62,204	71,108	2,865	3,275	5,209	5,960	65.962	21.70	20.20	Caked, friable, small
No. 5......	32,017	38,528	58,212	70,048	2,832	3,408	5,149	6,196	64.016	22.80	....	Caked, very friable, small
Peat......	76,564	99,230	139,204	180,416	4,333	5,615	7,860	10,209	27.304	55.00	....	Not caked

Euchéne now gives us a complete equation for the exchange in heat. The first member of this equation comprises two terms, viz., A = the heat furnished by the fuel, and B = the heat of formation of the volatile products which are formed with the liberation of heat.

Table XLIII gives the value of B.

The second member comprises seven terms, or C = the heat carried away by the smoke, D = the heat carried away by the volatile products of distillation, E = the heat retained by the red hot coke, F = the heat lost by radiation from the brick-work and mouthpieces, G = the heat of formation of the compounds of carbon which are formed with the absorption of heat, this value being found in Table LXIII, H = the heat carried away by the ash-pit refuse, and X = the heat of decomposition of the coal. This equation will therefore read

$$A + B = C + D + E + F + G + H + X \qquad (53)$$

and the value of X will have to be determined by difference. As already stated, the value of B and G are given in Table LXIII; therefore in order to find the value of X it will first be necessary to determine the value of the other terms of the equation.

Distillation proceeded slowly with Nos. 1 and 2; rather quickly with No. 3; quickly with Nos. 4 and 5; and rapidly with peat.

By comparing this equation with equation (52), it becomes evident that $x = X - B + G$, or the x of equation (52) is the actual heat of decomposition, while the X of equation (53) is a value comprised in x. If the coal is considered a ternary compound consisting of carbon, hydrogen, and oxygen, its decomposition may be supposed to occur in two stages, viz.: (1) The decomposition of this ternary substance into its elements of solid carbon, gaseous hydrogen, and gaseous oxygen, and X will be the heat expended in this first stage. (2) The formation of the volatile products, in which $-(B - G)$ will be the heat of formation, the negative sign indicating the liberation of heat. Then the actual heat x will equal $X - (B - G) = X - B + G$; consequently, X is only a portion of x.

Euchéne now determines the values of the terms in equation (53), and starts with the first members of the equation, or A and B. The determination of A, or the heat furnished by the fuel, is considered first. The coke employed for fuel had the following mean composition in percentages by weight.

Moisture	6.0 per cent.
Ash	10.0 per cent.
Carbon	81.5 per cent.
Hydrogen	0.5 per cent.
Sulphur	1.0 per cent.
Nitrogen	1.0 per cent.

This coke was measured in hectoliters weighing 40 kilos each (32 lb. per bushel).

The heat liberated when 1 kilo of carbon is converted into carbon dioxide equals 8080 calories; when 1 kilo of hydrogen is burned to water, liquid at 0°C., equals 34,462 calories; and when 1 kilo of sulphur is consumed this heat amounts to 2000 calories. EUCHÈNE includes the latent heat of vaporization of the water formed in the loss under the head of smoke, etc.

The term A is very simple for ordinary direct-fired furnaces, but for generator furnaces, where combustion is effected in two stages, we must differentiate between (1) the incomplete combustion of the coke fuel in the producer, resulting in the production of combustible gases, and (2) the combustion of these gases having essential heat, with the hot air from the recuperator.

It therefore follows that the heat furnished in the furnace is made up of (*a*) the heat supplied by the combustible gases, (*b*) the heat of combustion of these gases, and (*c*) the heat supplied by the hot air. The term B, or the heat of formation of the volatile products, will be taken from Table LXIII, it amounting to 29,243 calories per 100 kilos of coal No. 3, or 53,176 B.t.u. per 100 lb. of coal distilled.

The first term of the second member of the equation is C, or the heat lost in the smoke or chimney gases. The heat held by a gas at a given temperature is equal to the product of the volume of gas and the heat required to bring the unit volume of the gas to a given temperature. When the heat required to raise the temperature of a gas varies in proportion to the temperature reached, it is merely the product of the constant specific heat of the gas and the temperature; but when this heat does not so vary proportionately, the specific heat is no longer constant, but for any given temperature it may be found by an expression which will embody the results of three observations at different temperatures. The specific heat of the gas rarely remains constant at high temperatures.

In the sequel, only the heat ($=Q$) required to heat a gas to the temperature (*t*), or the heat possessed by the gas at that temperature, need be considered, and this amount will have been determined directly. Only two determinations will be required, viz., (1) that of the temperature of the smoke or flue gases, and (2) that of the heat required to raise the smoke to that temperature. The temperatures in EUCHÊNE's experiments were determined with a LECHATELIER pyrometer, and they are given in Table LXIV. The heat required to raise the gases to given temperatures is not proportional to the temperature, and it is found to vary according to the nature of the gas, and also according to the temperature.

TABLE LXIV.—HEAT WHICH RAISES THE NAMED GASES TO THE TEMPERATURE STATED

Temperature	1050°C. = 1922°F.		975°C. = 1787°F.		800°C. = 1472°F.		650°C. = 1202°F.		600°C. = 1112°F.	
Where observed	Temperature of chimney gases at the outlet of a Siemens furnace		Smoke or chimney gases from an ordinary furnace		Combustible producer gas		Volatile products issuing from the retort		Outlet of a Siemens recuperator	
	Calories per cbm.	B.t.u. per cu. ft.	Calories per cbm.	B.t.u. per cu. ft.	Calories per cbm.	B.t.u. per cu. ft.	Calories per cbm.	B.t.u. per cu. ft.	Calories per cbm.	B.t.u. per cu. ft.
Nitrogen, oxygen, hydrogen, carbon monoxide..	343	38.8	317	36.0	257	28.8	207	23.6	190	21.6
Steam (total heat)......	1,005	113.6	956	108.4	851	96.4	767	86.8	741	84.0
Carbon dioxide.........	606	68.8	550	62.4	426	48.4	329	37.2	298	33.6
Marsh gas.............	......	......	......	......	......	......	398	45.2	.....	.....
Sulphur dioxide.........	908	102.8	863	97.6	758	86.0	.....	.....	638	72.4
Benzene...............	......	......	......	......	......	.....	1287	145.6	.....	.....
Cyanogen, carbon disulphide, hydrogen sulphide, ammonia.							390	44.0	.....	.....

Tar (650°C. = 1202°F.) 423 calories per kilo, 768 B.t.u. per pound.

The second term of the second member of the equation, *D*, represents the heat lost through the volatile products of the distillation; these products are gas, tar, and water, and the values given in Table LXIII for coal No. 3 will be used in the following computations, but their temperatures must be determined. This temperature was observed on top of the charge at a distance of 1 ft. from the mouth of the retort, and was found to be 630°C. (1166°F.) at the beginning and 735°C. (1355°F.) at the close of the distillation period. But as the bulk of the volatile products is given off in the first 2 hours, their mean temperature, as they leave the retort, will be taken at 650°C. (1202°F.).

In the ascension pipe, at a point 8 in. from the mouthpiece, the temperature of these volatile products was found to have dropped to between 420° and 440°C. (788° and 824°F.), while at the hydraulic main the maximum was 80°C. (176°F.). Taking the composition of the volatile products as already given, and the heat required to raise them to 650°C. (1202°F.) as given in Table. LXIV, calculation will give the following values for the heat removed by the volatile products in the distillation of 100 kilos, or 100 lb. of type 3 coal:

	Calories	B.t.u.
Carbon dioxide	173.7	315.8
Carbon monoxide	519.8	944.8
Hydrogen	3,151.2	5,729.6
Marsh gas	4,149.2	7,544.0
Nitrogen	63.4	115.2
Benzene	380.2	691.2
Ethylene	556.8	1,012.0
Hydrogen sulph., ammonia, etc.	207.7	377.6
Gas total	9,202.0	16,730.2
Water	6,492.0	11,804.0
Tar	2,305.4	4,192.0
Total	17,999.4	32,726.2

For the other four types of coal, the total heat removed by the volatile products from 100 kilos or 100 lb. of coal, was:

	No. 1		No. 2		No. 4		No. 5	
	Calories	B.t.u.	Calories	B.t.u.	Calories	B.t.u.	Calories	B.t.u.
Gas	8,839.5	16,072	9,224.2	16,772	9,131.9	16,604	8,447.6	15,360
Water	4,373.1	7,952	5,310.9	9,656	8,220.6	14,946	9,407.4	17,104
Tar	1,903.5	3,460	2,144.6	3,899	2,664.9	4,852	2,707.2	4,922
Totals	15,116.1	27,484	16,679.7	30,327	20,017.4	36,402	20,562.2	37,396

An examination of these figures will show that (1) the hydrogen and marsh gas are responsible for the bulk of the heat carried away by the gas; and (2) that the heat carried away by the gas is nearly constant for all the types of coal, but that removed by the tar and water, especially the water, increases from type No. 1 to type No. 5. As the heat liberated and absorbed by the formation of the volatile products varies appreciably for the several types, it is clearly necessary, in drawing up an equation of the heats, to distinguish between these different types.

The next term, *E*, represents the heat lost through the red hot coke. When the couple of the LeChatelier pyrometer was inserted in the charge in the middle of the retort, the temperature ½ hour after charging was found to be 500°C. (932°F.), and it rose to 950°C. (1742°F.) at the end of the distilling period. Consequently, the coke must leave the retort at an average temperature of about 950°C. (1742°F.)—in some retorts, and in some parts of each retort, rather less or rather more, but this will be a fair average temperature. As the capacity for heat of carbon at high temperatures is given differently by different authorities, Euchène made fresh determinations for the purpose of this investigation. He found that the heat removed by 1 kilo of coke at 950°C. was 350 calories (636 B.t.u. for 1 lb. of coke at 1742°F.); hence it followed that the heats removed by the amounts of coke obtained from each of the five types of coal was:

	No. 1	No. 2	No. 3	No. 4	No. 5
Calories per kilo	26,481	25,573	24,585	23,087	22,406
B.t.u. per pound	48,120	46,152	44,674	41,952	40,714

The next term, *F*, comprises the heat lost through the brick-work, mouthpieces, grate, opening the retorts, etc. These losses were found to vary with the season, time of day, and state of the atmosphere, and were therefore not readily estimated. Existing data for the conductivity and radiating powers of brick-work at high temperatures were found to be unreliable, and Euchène therefore made a fresh determination of the coefficient of conductivity. The losses by radiation, etc., were thus found to amount, per setting per hour, to 54,824 calories (219,296 B.t.u.) with the ordinary setting of seven retorts; to 80,174 calories (320,696 B.t.u.) with a Siemens setting of eight retorts; and to 89,251 calories (357,000 B.t.u.) for a Siemens setting of nine retorts.

The last term, *H*, of the equation represents the loss of heat through the ashes, etc., and it may be taken from metallurgical observations at 400 calories per kilo (728 B.t.u. per pound). This figure is perhaps rather high for gas-works furnaces because it refers to molten slag, but the error thus introduced would be of no moment.

The several terms of the equation having been determined, they were applied in the cases of coals of types Nos. 2, 3, and 4, distilled in an ordinary setting of seven retorts, and in the case of coal No. 3 to distillation in a SIEMENS setting of eight retorts.

A summary of the data and calculations in the case of coal No. 3 distilled in an ordinary setting is given in the following:

COAL OF TYPE NO. 3

Heat produced and expended during distillation in an ordinary setting.

Coke consumed = 21.2 parts per 100 parts (by weight) of coal distilled.

Water evaporated from ash pan } = 6.05 parts per 100 parts (by weight) of coal distilled

The composition of the coke was:

	Per cent.
Moisture	6.0
Ash	10.0
Carbon	81.5
Hydrogen	0.5
Sulphur	1.0
Nitrogen	1.0
	100.0

The smoke or chimney gases consisted of 16 per cent. by volume of carbon dioxide (containing 8.582 per cent. of carbon), 4.5 per cent. of oxygen, and 79.5 per cent. of nitrogen, and their temperature was 975°C. (1787°F.).

EUCHÈNE discusses the results of the observations made on the distillation of coals Nos. 2, 3, and 4, in the ordinary setting. In the first place, he makes some comparisons of the three types of coals with reference to the heat of decomposition and the thermic reactions which occur in the retort, recapitulating the data given in Table LXV for these three coals.

TABLE LXV.—HEAT DATA IN CALORIES PER 100 KILOS, AND B.T.U. PER 100 LB.

Type	Heat liberated by the volatile products		Heat of formation of the carbon compounds		Heat carried off by the volatile products		Heat of the red-hot coke		X		Heat of decomposition	
	Cal.	B.t.u.	Cal.	B.t.u.	Cal.	B.t.u.	Cal.	B.t.u.	Cal.	B.t.u.	Cal.	B.t.u.
No. 2	26,228	47,688	2,282	4,148	16,680	30,328	25,573	46,496	22,707	41,284	−1,239	−2,252
No. 3	29,243	53,168	2,547	4,632	17,999	32,724	24,585	44,700	23,098	41,996	−3,598	−6,542
No. 4	34,216	62,212	2,865	5,208	20,017	36,398	23,087	41,976	25,000	45,456	−6,351	−11,620

The negative sign before the values under "Heat of Decomposition," or x, indicates that heat is liberated by the distillation of coal, but as the amount thus liberated is not large, it might be considered as falling within the limits of the probable errors in the observations. The liberation of heat here shown is, however, supported by MAHLER's results of 1893, he having made use of a totally different method of investigation.

MAHLER determined the calorific value of 100 kilos of coal of type No. 4, as well as the calorific value of all the products obtained on carbonizing it. The calorific value of the coal exceeded that of the resultant products by 25,483 calories (46,332 B.t.u. per 100 lb.), which quantity of heat had, therefore, disappeared, or had been liberated during carbonization, this being a further proof of the statement that the distillation or carbonization of coal liberates heat.

On comparing this figure with that found in EUCHÉNE's researches covering the same type of coal—viz., 6351 calories—it will be observed that the values both indicate a liberation of heat, and the difference of 19,000 calories, though apparently great, may be accounted for by the inevitable sources of error, especially as the errors would be in opposite senses in the two cases. For instance, in EUCHÉNE's research, the heat of decomposition (x) has been computed by taking the difference between the heat produced (m) and the heat consumed (M), thus $x = M - m$. Here x has a negative value, and sources of error would make (M) low in the same manner as (x). But in MAHLER's method, $x = C - c$, where (C) is the heat of combustion of the coal, and (c) the heat of combustion of the products; and (c) is likely to be too low, and consequently (x) too great.

It is to be observed in the next place that the heat liberated by the volatile products increases from 26,228 calories for coal No. 2 to 34,216 calories for coal No. 4, while the heat of decomposition ranges from 1239 to 6351 calories. Further, the sum of the heat carried off by the volatile products and of the red hot coke is nearly constant, viz., 42,253 calories for No. 2, 42,584 calories for No. 3, and 43,104 calories for No. 4. It follows that the heat of decomposition is characteristic of the different coals, just as is the proportion of volatile matter, but it is affected by the physical phenomena of distillation. Fusion takes place to an extent which varies with the type of coal, it being more marked with No. 2 than with No. 4, and it is likely that fusion absorbs a certain amount of heat.

Again, it is noticeable that the carbonization of coal liberates heat if it is assumed that there can be recovered (1) the heat carried off by the volatile products, (2) the heat of the hot coke, (3) the heat of the ashes, and (4) the heat of the chimney gases, also that radiation can be completely prevented. Therefore, if heat is once applied to start carbonization, it will proceed without the expenditure of fuel. In other words, carbonization of coal can take place without the expenditure of fuel, this observation elucidating the phenomenon of the heating up of coal, which EUCHÈNE attributes to oxidation. It is not, according to EUCHÈNE,[1] the sulphur content of the coal which determines the liability to oxidize, because coals which contain much sulphur, such as English coals, are less liable to spontaneous ignition than those which contain very little sulphur, such as Belgian coals. When the oxidation occurs in a small heap, the heat produced is dissipated, and heating of the coal does not occur; but when the coal pile exceeds a certain height, say 13 to 18 ft., and especially when the lumps are large and the intervening air spaces are considerable, the absorption of the enclosed air by the coal produces a little heat, the latter thus favoring oxidation of the coal. When this occurs, due to the feeble conductivity of the coal, distillation commences, and continues slowly until the volatile matter is completely separated, so that after a time, the conversion of the coal into coke may be almost as complete as in a retort.

Though complete recovery of heat, such as has been imagined in the case of the coal pile, cannot be effected industrially, it is desirable to consider under what conditions an approach to this state can be made. EUCHÈNE states that in the first place the average composition of the chimney gases—carbon dioxide 16 per cent., oxygen 4.5 per cent., and nitrogen 79.5 per cent.—shows an excess of 4.5 per cent. of oxygen, this being the normal state of affairs with ordinary furnaces. The chimney gases should theoretically consist of 20.5 per cent. of carbon dioxide and 79.5 per cent. of nitrogen. For the quantity of coke consumed, there should be produced 32,206 cubic meters (for 21.2 lb. of coke per 100 lb. of coal, 516.76 cu. ft.) of carbon dioxide and 124,895 cubic meters (2004 cu. ft.) of nitrogen, making a total volume of 157,101 cubic meters (2520.76 cu. ft.) in place of the 201,304 cubic meters (3230 cu. ft.) produced with the usual excess of air. The heat carried off by the 157,101 cubic meters would be 57,315 calories (104,208 B.t.u.) instead of 71,335 calories (129,934 B.t.u.) under the usual conditions. This excess of air therefore entails a loss of 14,000 calories (25,726 B.t.u.), or 9.7 per cent. of the heat furnished by the coke.

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[1] See Chap. II.

This important loss can be avoided only with difficulty, just as in other ordinary grates where the supply of air is not well under control. It is, however, possible to diminish the loss by altering the draft until incomplete combustion takes place in the bed of the fuel, thus forming a little carbon monoxide which is subsequently burnt by a little air admitted through holes in the furnace doors. Only a slight excess of air is required by this method of operation, which is practically combustion in two stages, and which is more thoroughly carried out in producer furnaces.

The water evaporated in the ash pan is responsible for a loss of 7196 calories (13,084 B.t.u.), or 5 per cent. of the heat furnished by the coke. This loss is unavoidable, because the furnace bars must be kept cool, and in producers the greater portion of this water is decomposed, and thus produces hydrogen. The use of red-hot coke for fuel would result in an economy of 8933 calories (16,240 B.t.u.), or 6.2 per cent. of the heat furnished by the coke. This economy is appreciable, but, according to EUCHÊNE, the practical difficulties of charging the furnace with hot coke, and the interference with the stoker's routine, would often make it impossible to secure even a part of this advantage.

The loss of heat by radiation is influenced by the quantity of coal carbonized in the setting. For instance, seven retorts carbonized 5600 kilos (12,320 lb.) of No. 2 coal in 24 hours, with a loss by radiation of 23,496 calories per 100 kilos (42,720 B.t.u. per 100 lb.); or 5950 kilos (13,090 lb.) of No. 3 coal, with a loss of 22,114 calories (40,208 B.t.u.); or 6300 kilos (13,860 lb.) of No. 4 coal, with a loss of 20,885 calories (37,972 B.t.u.). Therefore, if eight retorts were placed in a setting of the same size and surface, and 7200 kilos (15,840 lb.) of coal were distilled per 24 hours in them, the loss by radiation would be reduced to 18,260 calories per 100 kilos (33,200 B.t.u. per 100 lb.) or an economy of 5000 calories (9520 B.t.u.) as compared with the distillation of No. 2 coal quoted above, this being equivalent to a saving of 3.2 per cent. of the heat furnished by the coke fuel. For No. 3 coal the losses by radiation amounted to 12.52 per cent. of the total heat, and they were distributed among the top and sides of the setting, the mouthpieces, opening the retorts, etc. Economy at any one of these points would have but little effect, as the loss at each is small, and an appreciable effect on these losses can be made only by increasing the charge of coal per setting.

To sum up, the ordinary furnace and setting, as commonly used, appears susceptible of improvement being effected in the following directions:

On the excess of air	an economy of 9.50 per cent.
On water in ash pan	an economy of 4.90 per cent.
On red hot coke	an economy of 6.16 per cent.
On losses by radiation	an economy of 3.20 per cent.
Total	23.76 per cent.

Modern furnaces, in which combustion is divided into two stages, or incomplete combustion, with the formation of combustible gas, in the producer, followed by the combustion of this gas in the setting, have the advantages of avoiding the destructive combustion on the grate of the ordinary furnace; of placing the grate at a distance from the setting, and separating the work done by the grate from that done in the setting or furnace proper; of securing an easily regulated supply of air to the setting; and of utilizing the heat of the chimney gases.

These advantages were considered in connection with the SIEMENS setting, in which only the No. 3 type of coal was carbonized for the purposes of EUCHÈNE's investigations; the heat balance for this coal in an ordinary setting is given in Table LXVI.

TABLE LXVI.—HEAT BALANCE FOR 100 KILOS AND 100 LB. OF COAL OF TYPE NO. 3

HEAT PRODUCED

Term	Kilos	Calories	Pounds	B.t.u.
A. Heat from 21.2 kilos and 21.2 lb. of coke.				
Carbon	17.278× 8,080 =	139,606.240	17.278×14,692 =	253,848
Hydrogen	0.106×34,462 =	3,652.972	0.106×62,658 =	6,642
Sulphur	0.212× 2,000 =	424.000	0.212× 4,050 =	859
		143,683.000		261,349
B. Heat of the volatile products		29,243.000		53,176
		172,926.000		314,525

HEAT CONSUMED

Term	Cubic meters	Calories	Cubic feet	B.t.u.
C. Loss of heat through smoke at 975°C. (1787°F.)				
Carbon dioxide	32.213 × 549.700 =	17,707	516.76 × 62.4 =	32,246
Oxygen	9.058 × 317.153 =	2,875	145.79 × 36.0 =	5,248
Nitrogen	160.033 × 317.153 =	50,753	2,567.77 × 36.0 =	92,440
Water of the coke	1.582 × 956.224 =	1,513	25.42 × 108.4 =	2,756
Water of the ash pan	7.525 × 956.224 =	7,196	120.73 × 108.4 =	13,087
Water from hydrogen	1.186 × 956.224 =	1,139	19.06 × 108.4 =	2,066
Sulphur dioxide	0.148 × 863.000 =	128	2.38 × 97.6 =	232
		81,311		148,075
D. Volatile products at 650°C. (1202°F.)		17,999		32,726
E. Red hot coke at 950°C. (1742°F.)	70.243 × 350.000 =	24,585	70.243 × 636.0 =	44,674
F. Losses by radiation	$\frac{54,824 \times 24 \times 100}{5,950} =$	22,114		40,208
G. Formation of carbon compounds		2,547		4,631
H. Ash, etc. (15 per cent. of the coke)	3.18 × 400.000 =	1,272	3.18 × 728.0 =	2,315
X. By difference		23,098		41,896
		172,926		314,525

The heat of decomposition $x = X - B + G = 23,098 - 29,243 + 2,547 = -3,598$ cal.
$= 41,896 - 53,176 + 4,631 = -6,649$ B.t.u.

Table LXVII shows the heat balance in percentages for the three types of coal, Nos. 2, 3 and 4, distilled in the ordinary setting.

The heat balance for the ordinary setting was for coal distilled in seven retorts, while the balance for the No. 3 coal in a SIEMENS setting was for eight retorts. In this latter setting, EUCHÉNE found that the consumption of fuel varied with the season and the works, but the mean of several years with modern installations was 14.8 parts of coke per 100 parts (by weight) of coal distilled. In 24 hours, 7200 kilos (15,840 lb.) of coal were carbonized in the setting of eight retorts, and the water evaporated and carried into the furnace amounted to 40 per cent. of the weight of the coke, or 5.92 kilos per 100 kilos (5.92 lb. per 100 lb.) of coal carbonized,

TABLE LXVII.—HEAT BALANCE IN ORDINARY SETTING

HEAT PRODUCED

	No. 2	No. 3	No. 4
A. Heat from the coke	85.27	83.60	79.80
B. Heat from the volatile products	14.73	16.40	20.20
	100.00	100.00	100.00

MED

	48.30	47.20	45.20
	12.70	14.20	14.70
	14.40	13.20	13.60
	13.20	12.80	12.30
	9.40	10.40	11.80
as	1.30	1.50	1.70
	0.70	0.70	0.70
	100.00	100.00	100.00
kilos of coal			
alories	−1,239	−3,598	− 6,351
	−2,252	−6,649	−11,548

composition as that used in the ordinary

operation under three heads, viz., furnace, and (3) the recuperator or The average composition of the producer

	Per cent.
	6
	25
	8
	61

C. = 1472°F.

imney gases from the furnace consisted of:

	Per cent.
	18.37
	1.82
	79.81

1050°C. = 1922°F.

emperature was reduced to 600°C. (1112°F.) at the cuperator. In addition to the moisture in the to 1.104 cubic meters of vapor (17.72 cu. ft.), introduced into the producer 5.92 kilos, or 7.363 15 cu. ft.) of water evaporated under the grate, gas contained 5.803 cubic meters (93.19 cu. ft.) which the coke had yielded 0.828 cubic meter leaving 4.975 cubic meters (79.90 cu. ft.) of hydro- been produced by the decomposition of an equal vapor. Hence, of the water evaporated under the grate, 7.363 − 4.975 = 2.388 cubic meters (118.15 − 79.90 = 38.25 cu. ft.) remained as undecomposed water vapor.

The loss of heat through radiation, etc., from the furnace was found to amount to 55,933 calories (223,732 B.t.u.) per hour for the whole setting; as the setting carbonized 7200 kilos (15,840 lb.) of coal per 24 hours, the loss of heat from this cause amounted, per 100 kilos of coal carbonized, to

$$\frac{55{,}933 \times 24 \times 100}{7200} = 18{,}644 \text{ calories,}$$

or, per 100 lb. of coal, to

$$\frac{223{,}732 \times 24 \times 100}{15{,}840} = 33{,}900 \text{ B.t.u.}$$

The loss of heat through radiation, etc., from the recuperator was found to amount to 7158 calories per hour; therefore, the loss of heat from this cause, per 100 kilos of coal carbonized, amounted to

$$\frac{7158 \times 24 \times 100}{7200} = 2386 \text{ calories,}$$

or, per 100 lb. of coal, to

$$\frac{28{,}632 \times 24 \times 100}{15{,}840} = 4338 \text{ B.t.u.}$$

Table LXVIII shows the heat provided and that consumed in a **Siemens** setting of eight retorts, while Fig. 73 gives us a graphic comparison of the results recorded in Tables LXVI and LXVIII for the two types of setting.

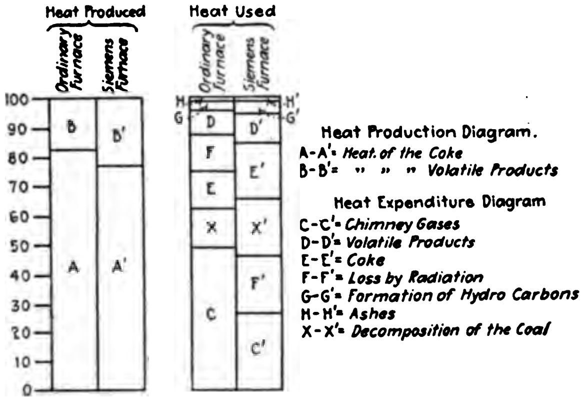

Fig. 73.

TABLE LXVIII.—HEAT BALANCE FOR 100 KILOS, AND FOR 100 LB. OF COAL NO. 3, DISTILLED IN A SIEMENS SETTING OF EIGHT RETORTS

PRODUCER

HEAT PRODUCED

Term	Kilos	Calories	Pounds	B.t.u.
Aa Heat from coke—				
C of the CO_2	2.335×8,080 =	18,858.720	2.335×14,692 =	34,306
C of the CO	9.728×2,473 =	24,057.344	9.728× 4,496 =	43,737
Sulphur	0.148×2,000 =	296.000	0.148× 4,050 =	599
		43,212.000		78,642
Difference undetermined		1,418.000		2,139
Total		44,630.000	Total	80,781

HEAT CONSUMED

	Cubic meters	Calories	Cubic feet	B.t.u.
C'- Heat of the combustible producer gas—				
Carbon dioxide	4.352× 425.942 =	1,854	69.89× 48.4	= 3,383
Carbon monoxide	18.135× 257.092 =	4,662	291.00× 28.8	= 8,381
Hydrogen	5.803× 257.092 =	1,492	93.55× 28.8	= 2,694
Nitrogen	44.249× 257.092 =	11,376	709.88× 28.8	= 20,444
Water (undecomposed)	2.388× 851.013 =	2,032	38.30× 96.4	= 3,692
Water of the coke	1.104× 851.013 =	940	17.72× 96.4	= 1,708
Sulphur dioxide	0.103× 758.000 =	78	1.66× 86.0	= 143
		22,434		40,445
Decomposition of water vapor	4.975×3,079.00 =	15,318	79.78×348.8	= 27,827
	Kilos		Pounds	
H- Ash-pit refuse, 20 per cent. of coke,	2.96 × 400.0 =	1,184	2.96×728.0	= 2,155
F'— Radiation—				
From surface of producer (7,661 cal. per hour) =		2,554	(30,644 B.t.u. per hour) =	4,643
From fire bars of producer (2,265 cal. per hour) =		754	(9,060 B.t.u. per hour) =	1,373
From gas conduit of producer (7,158 cal. per hour) =		2,386	(28,632 B.t.u. per hour) =	4,338
Total		44,630	Total	80,781

FURNACE

HEAT PRODUCED

	Calories		B.t.u.
C'- Heat introduced by hot producer gas	22,434		40,788
Ab- heat furnished by combustion of producer gas			
Cubic meters Kilos		Cubic feet Pounds	
CO = 18.133 = Carb. 9.728× 5.607 =	54,545	291 = Carb. 9.728×10.196 =	99,186
H_2 = 5.803 = 0.518×34.462 =	17,865	93 = 0.518×62.656 =	32,456
C'- Heat introduced by hot secondary air =	22,335		40,608
B- Heat furnished by volatile products =	29,243		53,176
Total	146,422	Total	266,214

TABLE LXVIII.—*Continued*

FURNACE

HEAT CONSUMED

	Cubic meters		Calories	Cubic feet		B.t.u.
C'—Heat of chimney gases at 1050°C. (1922°F.)						
CO_2 from producer..	4.352	× 606.081 =	13,629	70.0	× 68.68	= 24,862
CO_2 from CO.......	18.135	× 606.081 =		292.0		
Water of the coke....	1.104	× 1,005.372 =	9,344	17.7	× 114.00	= 16,997
Water undecomposed	2.388			38.3		
Water from the H_2..	5.803			93.1		
Nitrogen from producer.............	44.249	× 343.318 =	33,536	710.0	× 38.88	= 60,925
Nitrogen of sec. air..	45.022			722.0		
Nitrogen in excess...	8.410			135.0		
Oxygen in excess.....	2.235	× 343.318 =	767	35.8	× 38.88	= 1,392
Sulphur dioxide......	0.103	× 908.000 =	94	1.7	× 102.88	= 175
D—removed by volatile products at 650°C. (1202°F.)............		=	17,999			32,726
E—heat of red hot coke at 950°C. (1742°F.)		=	24,585			44,700
F^1—losses by radiation, etc..............			18,644			33,898
G—heat absorbed by formation of carbon compounds..........		=	2,547			4,631
X—by difference..............			25,277			45,908
Total..............			146,422	Total..............		266,214

Heat of decomposition $x = X - B + G = 25{,}277 - 29{,}243 + 2{,}547 = -1{,}419$ Cal.
Heat of decomposition $x = X - B + G = 45{,}908 - 53{,}176 + 4{,}631 = -2{,}637$ B.t.u.

RECUPERATOR

HEAT INTRODUCED

	Calories	B.t.u.
C'—heat of chimney gases at 1050°C. (1922°F.)	= 57,370	 104,128

HEAT CONSUMED

	Cubic meters		Calories	Cubic feet		B.t.u.
C—heat of chimney gases at 600°C. (1112°F.)						
CO_2 from producer......	4.352	× 297 951 =	6,700	70.0	× 33.76	= 12,221
CO_2 from CO...........	18.135			292.0		
Water of the coke.......	1.104	× 740.796 =	6,866	17.7	× 83.92	= 12,512
Water undecomposed....	2.388			38.3		
Water from H_2.........	5.803			93.1		
Nitrogen from producer..	44.249	× 190.131 =	18,572	710.0	× 21.54	= 33,753
Nitrogen of sec. air......	45.022			722.0		
Nitrogen in excess.......	8.410			135.0		
Oxygen in excess........	2.235	× 190.131 =	425	35.8	× 21.54	= 771
Sulphur dioxide.........	0.103	× 638.000 =	66	1.7	× 72.28	= 124
F^1—Losses by radiation, etc................			2,386			4,338
C—Heat carried off by secondary air, by difference..............		=	22,355			40,409
Total..............			57,370			104,128

TABLE LXVIII.—*Continued*

RECAPITULATION

HEAT PRODUCED

	Kilos	Calories	Pounds	B.t.u.
A—Heat from coke				
Carbon	12.062 × 8,080	= 100,307	12.062 × 14,692	= 182,451
Hydrogen	0.074 × 34,462		0.074 × 62,656	
Sulphur	0.148 × 2,000		0.148 × 4,050	
B—Heat furnished by volatile products		29,243		53,176
Total		129,550	Total	235,627

HEAT CONSUMED

		Calories		B.t.u.
C—Heat of chimney gases at 600°C. (1112°F.)		32,649		59,381
D—Removed by volatile products at 650°C. (1202°F.)		17,999		32,726
E—Heat of red hot coke at 950°C. (1742°F)		24,585		44,700
F—Losses of radiation.	F^1 Producer.. 5,694	= 26,724	10,354	= 48,590
	F^2 Furnace .. 18,644		33,898	
	F^3 Recup.... 2,386		4,338	
G—Formation of carbon compounds		2,547		4,631
H—Ash-pit refuse		1,184		2,155
X—By difference		25,277		45,908
		130,965		238,091
Various, undetermined		1,415		2,464
Total		129,550	Total	235,627

Heat of decomposition $x = X - B + G = 25,277 - 29,243 + 2,547 = -1,419$ cal.
Heat of decomposition $x = X - B + G = 45,908 - 53,176 + 4,631 = -2,637$ B.t.u.

TABLE LXIX.—NO. 3 COAL.—HEAT BALANCE FOR SIEMENS SETTING IN PERCENTAGES

Heat produced		Heat consumed	
I. PRODUCER			
Heat from the coke	100.0	Heat of producer gas	50.4
		Decomposition of water	34.2
		Losses through radiation	12.8
		Ashes, etc	2.6

II. FURNACE

Combustion of producer gas	49.4	Heat of chimney gases at 1050°C. (1922°F.)	39.2
Heat furnished by volatile products	20.0	Decomposition of coal	17.3
Heat furnished by hot producer gas	15.3	Heat of the coke	16.8
Heat introduced by secondary air	15.3	Losses through radiation	12.7
		Heat removed by volatile products	12.3
		Formation of carbon compounds	1.7

III. RECUPERATOR

Heat of chimney gases at 1050°C. (1922°F.)	100.0	Heat of chimney at 600°C. (1112°F.)	56.9
		Heat removed by secondary air	38.9
		Losses by radiation	4.2

GENERAL RECAPITULATION

Heat of coke	77.4	Heat removed by chimney gas at 600°C	25.2
Heat furnished by volatile products	22.6	Losses by radiation	19.9
		X	19.5
		Heat removed by the coke.	18.9
		Heat removed by volatile products	13.8
		Formation of carbon compounds	1.9
		Ashes, etc	0.8

An examination of these tables shows that the differences between the two types of settings are chiefly in the losses due to the smoke of the chimney gases, and though some of the other losses appear higher in the SIEMENS setting, they are compared with a smaller consumption of fuel. The losses through radiation are, however, really greater with the SIEMENS setting, because of the producer and recuperator, which are absent with the ordinary setting.

On comparing the heat carried off by the chimney gases, the following differences are apparent:

	Temperature of the chimney gases	Heat lost through chimney gases	Volume of chimney gases
Ordinary setting	975°C.	81,311 calories	212 cubic meters
SIEMENS setting	600°C.	32,649 calories	132 cubic meters
Differences	375°C.	48,662 calories	80 cubic meters

These figures show that fuel economy is not simply due to the difference of 375°C. (707°F.) in the temperature, but also to the smaller volume of the chimney gases. The heating of the air by these gases in the first place is responsible for an economy of fuel, which reacts on, and diminishes, the volume of the gases, thereby inducing a further economy. Although these are the two chief causes of economy, there are also several subsidiary ones. The excess of air in the ordinary setting, which is equivalent to an excess of oxygen amounting to 4.5 per cent., corresponds to a loss of heat of 14,000 calories, or 9.7 per cent. In the SIEMENS setting this excess air may be entirely avoided, but in practice the regulation of the air supply is such that a mean of 2 per cent. of oxygen is allowed in the chimney gases. This 2 per cent. of oxygen, however, has not the same significance as in the gases from an ordinary furnace because it is referred to a smaller volume of gases, and because the surplus volume leaves the setting ultimately at 600°C. (1112°F.) instead of 975°C. (1787°F.).

Again, with reference to the temperature of the chimney gases at the outlet of the furnace proper, 1050°C. (1922°F.) in the SIEMENS compares with 975°C. (1787°F.) in the ordinary setting. When the temperatures are observed at different parts of the settings, as shown in Figs. 74 and 75, we find the following results, Table LXX.

TABLE LXX.—TEMPERATURE IN SETTINGS

Setting	°C. at points				°F. at points			
	A	B	C	D	A	B	C	D
SIEMENS	1250	1120	1080	1050	2282	2048	1976	1922
	A′	B′	C′		A′	B′	C′	
Ordinary	1250	1070	975	......	2282	1958	1787	

If the temperature at that point in the ordinary furnace which corresponds with the point *A* in the SIEMENS furnace, viz., near the grate below the middle retort, is determined, it will be found to be 1350°C. (2462°F.), or even 1400°C. (2552°F.). The fall in temperature in the SIEMENS furnace will be from 1250° to 1050°C. =200°C. (2282° to 1922° = 360°F.); and in the ordinary furnace, from at least 1350° to 975° = 375°C. (2462° to 1787° = 675°F.). The fall in temperature is equivalent to a thermic efficiency of 13.1 per cent. in the case of the SIEMENS, and 23.1 per cent. in that of the ordinary setting, which shows that, from the thermic standpoint exclusively, the ordinary setting utilizes the heat to better advantage than the SIEMENS setting, disregarding the recuperator of the latter. Considered from the gas-makers' standpoint, such a great fall of temperature must be avoided, because all the retorts in the setting should be charged alike, and the last should be heated as highly as the first. The SIEMENS setting most nearly gives this ideal condition of heating, and the defect indicated is not peculiar to the ordinary setting; it occurs in most furnaces in which the stream of gases continuously follows the same course, as in those regenerative furnaces and recuperative settings in which the stream of gas ascends in the middle and descends at the sides.

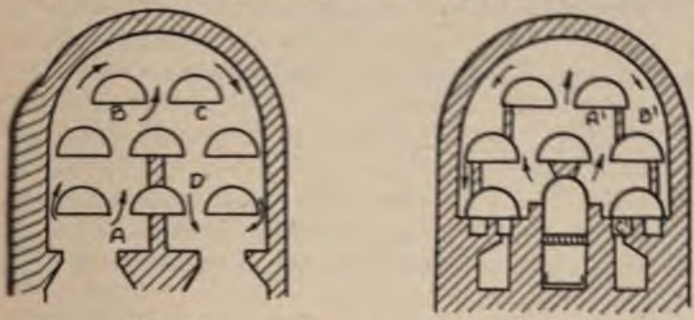

FIG. 74. FIG. 75.

FIGS. 74 AND 75.—Distribution of heat in settings.

In these cases the fall in temperature is less than with the ordinary setting, but it is greater than with the SIEMENS setting, because the latter admits of reversing the direction of the stream of gases, so that it is sent equally to the right and to the left. The more uniform heating which the SIEMENS alternating furnace affords presents the means of working off larger charges of coal in a proper manner, because all the retorts are equally hot; in the ordinary setting the first retort is very hot, and if the charges are regulated according to what this retort will work off, the charges in the other retorts will be only partially carbonized; it being practically impossible to alter the weight of the charge to suit each retort in a setting. Hence it has been found that a SIEMENS setting of eight retorts carbonizes 7200 kilos (15,840 lb.) per 24 hours, against about 5800 kilos (12,760 lb.) with an ordinary setting of seven retorts. Moreover, the number of retorts in a SIEMENS setting may be increased to nine, or even more, with advantage. Again, a great fall of temperature in the setting is fatal to the durability of the retort, while the grate of the ordinary setting also becomes rapidly corroded. Consequently, while the average life of retorts in ordinary settings is about 20 months, and the grate in that time requires repairs two or three times, the average life of retorts in the SIEMENS setting is 4 to 5 years, with slight repairs and cleaning of the recuperators at intervals of 20 months. Every means which, like the SIEMENS alternating furnace, helps to reduce the fall of temperature in the furnace chamber, is favorable to the carbonization of the coal, because it makes larger charges feasible, and prolongs the life of the retorts.

If the observation of the fall in temperature is extended to the exit of the recuperator of the SIEMENS setting, it will be found that the fall is from 1250° to 600°C., or 650°C. (2282° to 1112° = 1170°F.), against 375°C. (707°F.) observed in the ordinary setting. The thermic efficiency is 43.3 per cent. for the SIEMENS setting as against 23.1 per cent. for the ordinary, and this is the characteristic feature of the recuperative setting. On the other hand, the total losses from radiation are somewhat increased with the latter, but this is a slight disadvantage compared with the many advantages which recuperation affords. The heat of decomposition of coal of type No. 3 was found to be only −1419 calories (− 2637 B.t.u.) in the trials with the SIEMENS setting, against − 3598 calories (− 6649 B.t.u.) with the ordinary setting; this difference is very small when all the possible sources of error are considered.

EUCHÈNE states that in the producer the aim should be to secure gas as rich as possible in combustible constituents; theoretically, the producer gas should consist of 34.7 per cent. of carbon monoxide and 65.3 per cent. of nitrogen. The composition which has been taken as a normal average by EUCHÈNE is 6 per cent. of carbon dioxide, 25 per cent. of carbon monoxide, 8 per cent.

of hydrogen, and 61 per cent. of nitrogen. The gas, as it leaves the producer, would actually contain only 5 per cent. of carbon dioxide, but a slight combustion of carbon monoxide occurs in the gas nozzles leading to the combustion chamber. The sum of the combustible constituents is 25 + 8 = 33 per cent., which closely approximates the theoretical 34.7 per cent. of carbon monoxide, and hydrogen, volume for volume, has nearly the same calorific value as carbon monoxide. Thus the calorific power of the mixture actually obtained is practically the same as that of the theoretically perfect producer gas, but it should not be forgotten that 1.14 of the 8 per cent. of hydrogen is the hydrogen of the coke, and the remaining 6.86 per cent. comes only from the decomposition of water. Consequently, properly speaking, only 31.5 to 32 per cent. of the mixture consists of combustible constituents derived from the gasification process, and carbon dioxide has taken the place of the balance which should theoretically be present. But theoretical reactions cannot be secured in an industrial operation, and a small amount of carbon dioxide escapes the decomposition by which it would yield carbon monoxide. Furthermore the water in the pan beneath the grate bars, and which is required to help keep them cool, tends to reduce the temperature of the fuel and to impede the formation of carbon monoxide; while the decomposition of the water vapor by carbon yields hydrogen and carbon dioxide rather than hydrogen and carbon monoxide unless there is an excess of heat available to furnish the greater amount of heat required for the latter reaction.

The best conceivable conditions would be the evaporation of just sufficient water to keep the grate bars cool, and the avoidance of any surplus water vapor passing into the fuel to lower its temperature. In practice some water vapor passes undecomposed through the producer and leaves the recuperator at 600°C. (1112°F.), thereby carrying off 1769 calories. The amount will vary with the section of the producer and grate, being less where they are large, and frequent cleaning of the grate bars is unnecessary.

Though hydrogen and carbon monoxide have, volume for volume, nearly the same calorific power, the heat carried off by the products of their combustion differ greatly, being, at 600°C. (1112°F.), 298 calories for carbonic acid (542 B.t.u. per pound) and 741 calories (1347 B.t.u. per pound) for water vapor; hence it would appear to be more advantageous to burn carbon monoxide than hydrogen. But, on the other hand, the hydrogen and its attendant carbon monoxide are produced without the aid of primary air, though double the amount of secondary air is subsequently required, since a given weight of carbon always requires the same volume of air for its complete combustion. Where the secondary air supply only is heated, the decomposition of water vapor increases the heat recovered by recuperation.

Euchène states that means may be taken to limit the amount of carbon dioxide formed in the producer, as for example, the maintenance of a constant and sufficient depth (not less than 4 ft.) of coke in the producer, and the adoption of large producers in which the gas is brought into contact with the walls as little as possible. Euchène, by analyses of samples of gas drawn from different points in a producer, showed that carbon dioxide is formed first, carbon monoxide being a subsequent product.

At a distance of 12 in. above the grate bars, the gas contained from 3 to 8 per cent. of carbon dioxide and from 9.5 to 16.5 per cent. of oxygen, but no carbon monoxide. At a point from 24 to 28 in. above the grate bars, there was from 5 to 10 per cent. of carbon dioxide and from 11 to 25 per cent. of carbon monoxide. At a point 40 in. above the grate, the carbon dioxide had been reduced to from 2 to 4 per cent., and the carbon monoxide had been increased to from 26 to 28 per cent. In view of these facts it may safely be assumed that the transformation of carbon dioxide into carbon monoxide may be regarded as being first complete at about 40 in. from the grate.

The producer gas emerges from the producer at a temperature of about 800°C. (1472°F.), and it should be admitted into the combustion chamber as quickly as possible in order that there may be no loss of sensible heat. In the early Siemens furnaces, due to a long syphon passage, the gases were cooled to about 400°C. (752°F.), thereby entailing a loss equal to about 10 per cent. of the calorific value of the fuel, but in modern constructions this loss is avoided. The losses due to radiation amount to 12.8 per cent. of the heat liberated in the producer, or to 4.7 per cent. of the total calorific value of the fuel. The feeding of the producer with red-hot coke would result in an economy of 5.9 per cent. of the coke consumed, but, as already pointed out in the case of the ordinary setting, this plan is troublesome, and at times impracticable.

Referring to the furnace of the Siemens setting, it may be stated that the utility of the alternating procedure and the ease of regulating the secondary air supply are approved facts, but Euchène further states that the theory of recuperation may be considered with advantage. In the ordinary combustion of carbon to carbon dioxide, the chimney gases have a volume equal to that of the air required for combustion. When the combustion is effected in two stages, as in regenerative settings, the volume of primary air required is equal to that of secondary air and is half of the volume of the chimney gases produced. Consequently, when the secondary air only is heated, since there is double its volume of chimney gases, only half of their heating value is utilized. The heat in the recuperator is distributed in the following manner:

	Calories	B.t.u.		Calories	B.t.u.
Heat furnished by the chimney gases at 1050°C. = 1922°F.	57,370	104,128	Heat carried off by the chimney gases at 600°C. = 1112°F....	32,649	59,381
			Radiation..........	2,386	4,338
			Difference = the heat recovered by the secondary air......	22,335	40,409
	57,370	104,128		57,370	104,128

In systems of recuperation other than that of the Siemens alternating one, the heat is transmitted from the gases to the air through thin firebrick walls, but this class of recuperator was not investigated by Euchène; he does, however, point out that the alternating system precludes any admixture of the streams of gases and air, and affords uniformity in the heating of the retorts.

As the chimney gases leave the recuperator carrying 32,649 calories (59,381 B.t.u.), it is very important to determine if a portion of this heat can be utilized. A temperature of 150°C. (302°F.) is sufficient to ensure chimney draft, and this would necessitate 11,376 calories (20,684 B.t.u.) passing away with the chimney gases, leaving a balance of 21,273 calories (38,697 B.t.u.). The utilization of this amount of heat would be equivalent to a saving of 21.2 per cent. of the heating value of the coke. Heretofore, the utilization of this heat has been seldom attempted, owing to a fear of complicating the operation of the apparatus,[1] and although economy of fuel is always a desirable achievement, it is not the chief object of gas-works management.

The first feasible means of utilizing this waste heat consists in cooling the chimney gases from 600° to 150°C. (1112° to 302°F.) by continuous recuperation, the primary air being heated after the secondary air. The second scheme suggested by EUCHÉNE is to modify the SIEMENS alternating system so that both secondary and primary air are heated to a uniform temperature of about 1000°C. (1832°F.), and the chimney gases cooled to 150°C. (302°F.). EUCHÉNE deems this a rational scheme, but states that in order to carry it out, reversing or alternating of the primary air and the use of a steam injector, as well as of the secondary air and gases, would have to be arranged for, and in consequence of this arrangement, the SIEMENS system would become much more complicated than now.

EUCHÉNE next discusses the distribution of the heat furnished by the furnaces at the different stages of carbonization, it having been taken for granted hitherto that all the reactions occur uniformly and constantly. He states that it is difficult to locate the causes of variations which have their origin outside the retort, but that in the ordinary furnace the frequency of cleaning the grate has considerable effect; even when clinker is removed as frequently as once in 4 hours, the chimney gases will show great variations in composition. For instance, immediately after cleaning, these gases may contain 18 per cent. of carbon dioxide and 2 per cent. of carbon monoxide, while, just before cleaning, these gases may contain 14 per cent. of carbon dioxide and 6 per cent. of oxygen, the residue in both cases being nitrogen. These variations may be reduced by controlling the orifices in the furnace door. In regenerative settings, however, if the producers are of adequate size, only insignificant variations will be found, even though the fires are cleaned but once in 24 hours. The exterior of the retort does not vary greatly in temperature with the interior conditions, for it is constantly in contact with the hot gases and the hot firebrick supports in the setting.

The interior of the retort is, however, subject to some great variations. As a first step toward their investigation, EUCHÉNE made an attempt to determine the temperature at different stages during the period of carbonization. In order to avoid exposing the couple of the **Le Chatelier** pyrometer to the hydrogen sulphide in the retort, two iron tubes, plugged at each end, were used within the retort; one tube was plunged in the charge, and the other above it. Observations were made at distances of 1.3 meters (4 ft. 8 in.) and 0.45 meter (18 in.) from the mouth of the retort, the results secured being shown in Diagrams V and VI. The temperatures above the charge gradually rise as carbonization proceeds; but inside the charge these temperatures are more irregular, this irregularity being due to fluctuations in the course of the changes taking place, as well as to a want of homogeneity of the charge. We find that the temperatures increase away from the mouthpiece, and therefore Euchène assumed 950°C. (1742°F.) to be the average temperature of the red hot coke. The curves

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[1] See Waste Heat Boilers, Chap. XIV.

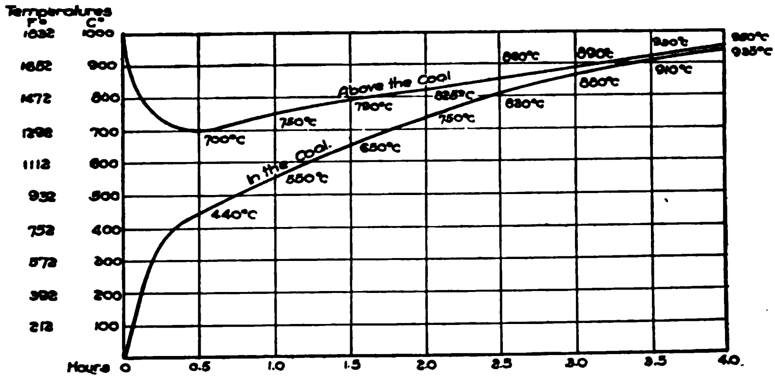

Diagram V.—Temperatures 4 ft. 8 in. from mouth-piece.

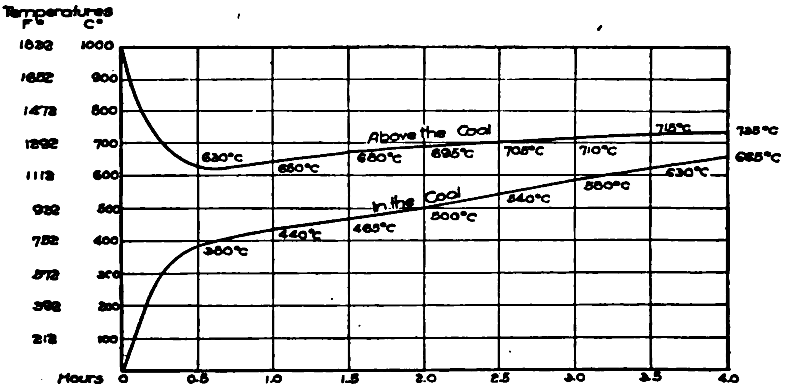

Diagram VI.—Temperatures 18 in. from mouth-piece.

of the diagrams indicate that the charge was carbonized under the best conditions between temperatures of 500° and 700°C. (932°-1292°F.); it is, however, necessary that a higher temperature than this should be attained, toward the end of the carbonizing period, in order that a quantity of heat may be stored within the retort, which heat will be given up to the next charge, and also that the interior of the charge may be properly carbonized.

With a carbonization period of 4 hours in a setting of seven retorts, the percentage of the total volume of gas evolved during each hour varied in accordance with the figures given in Table LXXI.

TABLE LXXI.—PERCENTAGE OF THE TOTAL VOLUME OF GAS EVOLVED

Type of coal	No. 1	No. 2	No. 3	No. 4	No. 5
In first hour	24.9	25.0	24.7	24.1	23.4
In second hour	29.9	28.4	29.2	29.6	26.9
In third hour	28.8	28.6	29.8	29.4	29.0
In fourth hour	16.4	18.0	16.3	16.9	20.7

The retorts, in succession, were charged as rapidly as possible, and the make from the whole setting was taken for calculating the yields for each hour. In order to make an exact comparison, it would have been necessary to carbonize exactly the same weight of each type of coal; but as some types require more heat for their proper carbonization than others, uniformity of conditions would be difficult of attainment, and EUCHÈNE states that the simplest plan to follow is to regulate the weight of charge so as to attain as nearly as possible uniform carbonization temperatures.

The proportion of carbon dioxide and nitrogen produced in the gas at the different periods may be taken as nearly constant, the percentage by volume of the remaining constituents of the gas each hour being shown in Table LXXII.

TABLE LXXII.—PERCENTAGE OF GAS CONSTITUENTS EACH HOUR

Hour	Carbon dioxide	Hydrogen	Marsh gas	Benzene	Ethylene
First hour	3.08	41.60	36.98	1.39	6.42
Second hour	1.90	49.72	32.08	1.24	3.46
Third hour	1.40	57.52	25.98	0.84	1.72
Fourth hour	1.08	58.64	23.72	0.25	1.12

coking operations was first mentioned by CLAYTON, in England, in 1737, and in 1781 the EARL OF DUNDONALD secured a patent for making coke in bee-hive ovens, his process also covering the production of tar, pitch, essential oils, etc., but not until 1792 do we find that any successful experiments were made in the production of gas from coal, the successful operator having been WM. MURDOCH, the father of the present-day coal-gas industry, the results of his labors having made it possible to light the streets of London in 1812. The first rectangular or retort ovens were built in Germany in 1830, and 5 years later, or in 1835, the first retort coke, with its accompanying gas, was made by WILLIAM FIRMSTONE at the Mary Ann Furnace in Pennsylvania. The first coke ovens in the Connellsville region are supposed to have been built in 1841, while in 1857 APPOLT built the first type of closed retort oven constructed to utilize the gas produced for its own heating. This was followed by COPPÉE, who built his first Belgian oven, using crushed coal, in 1861, and in 1893 the first battery of SEMET-SOLVAY ovens was erected in Syracuse, N. Y., soon to be followed by ovens of the OTTO-HOFFMAN, KOPPERS, and COLLINS types.

Mound Coke.[1]—When coal is coked in mounds, Fig. 76, it is first necessary to level off the coking ground and surface it with coal dust, after which the coal is arranged in heaps or pits, these heaps being provided with longitudinal, transverse and vertical flues, sufficient wood being distributed in these channels to ignite the entire mass. The base of this mound is made 14 ft. wide, and the coal is spread over same to a depth of 18 in., as shown at (*A*). The flues are arranged and constructed on this base in the manner shown in the plan, and the coal is piled up in the manner as shown at (*B*) in the section. The material of which these flues are made consists of refuse coke and lump coal, the whole being covered with billets of wood. When the heap has been constructed and ready for coking, fire is applied at the base of the vertical flues (*C*), the kindling wood at each alternate flue being lighted in turn. As the process of coking advances, the fire extends in every direction until the entire mass is ablaze. This method of coking requires considerable management in diffusing the fire evenly through the mass, in preventing the waste of coke by the access of too much air at any one place, and in banking up the heaps with fine dust as the coking operation progresses from the base to the top. When all of the gaseous matter has been burned off, the heap is carefully closed with dust or "duff" and nearly smothered out in this way. Finally, a small quantity of water is applied through the vertical flues, this water being quickly converted into steam and permeating the entire mass; if this water is properly applied, the coke will contain a minimum percentage of moisture. The state of the weather exercises quite some influence on the time of coking, this period being from 5 to 8 days.

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[1] Report L. Sec. Penn. Geological Survey, page 122.

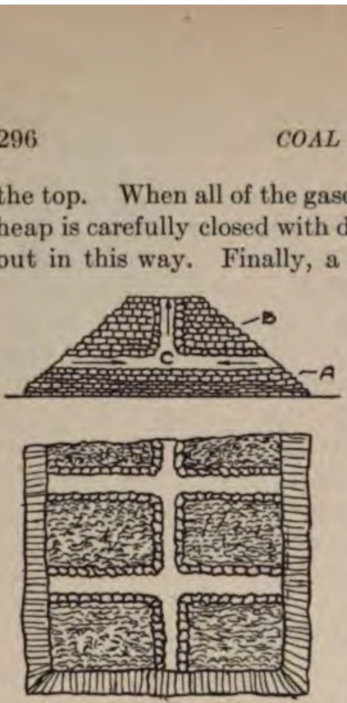

Fig. 76.—Coking mound.

Coke produced in this manner made an excellent fuel, but it was not always uniform, and the yield was small, because the process is wasteful.

Bee-hive Ovens.—The bee-hive oven is illustrated in Fig. 77, and it consists of a circular, dome-like chamber, lined with firebrick, the bottom, or floor, being constructed of flat tiles; the top is provided with an opening for charging the coal into the chamber, as well as for the escape of the products of combustion; the side of the chamber is provided with an arched door through which the air required for combustion is admitted, and through which the coke is watered and also drawn when the process is complete. These ovens are arranged in single stacks or banks, as well as in

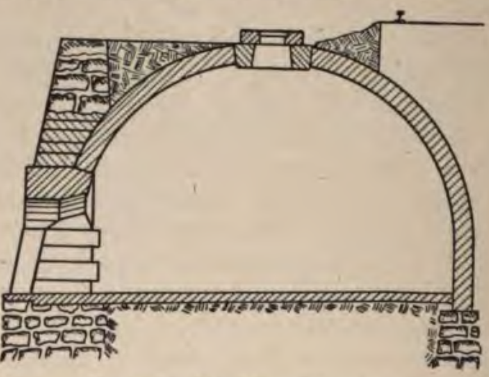
Fig. 77.—Bee-hive coke oven.

double rows, the ovens being located in a staggered line, Fig. 78, or back to back, Fig. 79, in the double-row construction.

The amount of coal charged into these ovens is dependent upon the time set for the coking operation, 48, 72 or 96 hours; the coal is delivered to the oven by a larry car running on the tracks above the oven, from whence it is discharged into the chamber through the opening in the top, the coal forming a conical pile within the chamber, this pile being leveled by a manual scraper. After the coal has been charged into the oven, the door is bricked up, an opening 2 or 3 in. deep being allowed at the top; the heat stored in the walls of the oven by the previous charge, or by preliminary heating, is sufficient to raise the temperature to such a degree as to cause the volatile matter liberated from the coal to catch fire. Under these conditions it will be found that the coking process will proceed from the top downward, while the required heat for the coking operation, and which is produced by the burning of the volatile matter escaping from the coal, is regulated by gradually closing up the opening left above the brick-work of the door, in a ratio dependent upon the decrease in amount of volatile produced; when no more volatile matter is expelled from the coal, the coking process will be found completed. At this stage the brick-work is removed from the door, and the coke is watered by means of a water spray, after which the coke is withdrawn from the oven.

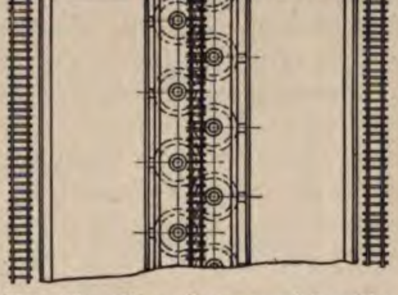
FIG. 78.—Plan of staggered bee-hive ovens.

The bee-hive oven is also arranged for mechanical drawing of the coke, but this latter arrangement usually breaks up the coke to a greater degree than hand-drawing, and thus produces a larger percentage of breeze and small coke. Mechanical drawing also requires that the door be made of greater dimensions, which enlargement makes draft regulation more difficult, and thus leads to the burning of quite an amount of coke. It is a practice at some plants to water the charge some time in advance of the time for drawing, and this practice usually leads to a rapid cooling down of the oven, thus reducing the coke yield, because this cooling down necessitates a longer time for igniting a subsequent charge.

Leveling the coal in the oven is of vital importance, but very few bee-hive oven operators pay sufficient attention to this detail; if the coal should not be level, or not of even depth, different portions of the charge become coked in different lengths of time, and this exposes the upper surface of the coke to a prolonged action of the air, as air is admitted until all volatile is expelled and burned; this condition leads to a loss of coke if the process is continued until all the coal is coked, or if the process should be stopped before all of the coal is coked, "black butts," or coal not completely coked, will result at some spots. A mechanical leveling device has recently been tried in the Connellsville region, and it is expected that this machine will assist in removing these difficulties.

The bee-hive oven operation is started by building a wood and coal fire within the oven, after which the door is closed with brick, an opening being provided for admitting sufficient air for proper combustion. When the ovens have been sufficiently heated, or when the firebrick lining has absorbed sufficient heat, the brick in the door is removed, and the oven chamber is cleaned. The oven is now prepared to receive its coal charge by again bricking up the door, leaving an opening of a size to permit the attendant to insert his scraper for leveling the charge, and a small amount of coal is charged and burned off, after which the full charge is admitted from the larry cars.

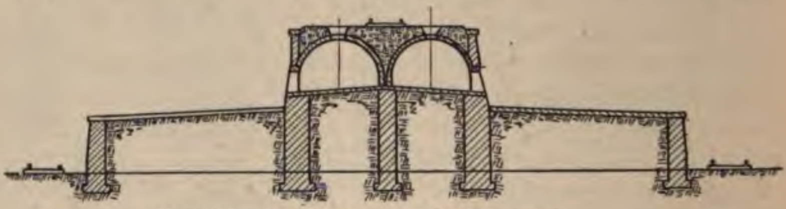

FIG. 79.—Bee-hive ovens, back to back.

Bee-hive ovens are the subject of great waste, not only in the loss of valuable by-products, but also in the amount of coke burned and low yield. A. W. BELDEN[1] states that "the handling of the modern bee-hive oven, as practised in all parts of the country, undoubtedly gives a lower yield of coke than might be obtained, but the matter of increased yield, even when brought to the attention of those in authority, is most often passed over with the remark, 'We are doing well enough and making money, so why should we make any change?' One reason why it is so hard to impress the importance of greater efficiency on the mind of the coke manufacturer is that he does not know what yield he is getting. He is satisfied to use figures that show the amount of coke produced if they come anywhere near what he thinks he ought to obtain. The 1911 returns to the division of mineral statistics of the United Stated Geological Survey show the average yield of coke in bee-hive ovens for the country to be 64.7 per cent., but if figures showing the actual tonnages of coal charged and coke produced were obtainable, the yield would probably more nearly approach that in mounds or piles, namely, 59 per cent. There is no doubt that with proper supervision of the burning of coke this figure could be increased 3 to 5 per cent. A conservative estimate of the direct loss from the 27,703,644 tons of coke produced in this country in bee-hive ovens in 1911 is 1,154,318 tons, worth over $3,266,350. The gratifying increase in by-product coke production, the output being 22.07 per cent. of the total production in 1911, shows that the country is waking up to the necessity of curtailing this enormous waste. That the importance of greater efficiency is being appreciated, and that a still more rapid increase in the number of by-product ovens may be expected, is shown by the large number of inquiries now in the hands of the by-product oven builders. The fact that the country's best coking coals are rapidly becoming exhausted makes the scientific study of the process of coking more imperative, and it is only the by-product oven that can give the proper answer to the many questions asked."

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[1] Tech. Paper 50, Metallurgical Coke, 1913, Bureau of Mines, A. W. BELDEN.

The year 1913 showed the production of 46,299,530 short tons of coke, valued at $128,922,273; of this amount 33,584,830 tons, or 72.5 per cent., were produced in the bee-hive oven, and 12,714,700 tons were produced in the by-product oven, an increase of about 5.5 per cent. in by-product coke over that produced in 1911. The increase in by-product oven coke in 1913 was 1,599,536 short tons over that produced in 1912, or 14.4 per cent., while the increase in bee-hive production amounted to 716,395 tons, or only 2.2 per cent.

Longitudinal Ovens.—The longitudinal coke oven was introduced in 1906, and its construction was due to an attempt to offset a scarcity of labor, with the expectation of using less skillful labor, its objective being the desire to make the coking operation independent of large labor forces and to thus decrease the actual cost of the coking operation. This oven is illustrated in Fig. 80, and it consists of a long, narrow, rectangular chamber, so proportioned that the discharging end is somewhat larger in cross-section than the opposite end, the bottom of the chamber being constructed of flat tiles, while the roof has a sort of barrel shape, sloping toward both ends; the roof is provided with a central opening, or "trunnel head," the ends of the chamber being supplied with doors. These ovens are built in blocks, the chambers lying side by side, and when in operation they are charged in a manner similar to the bee-hive oven, in that the coal is admitted through the trunnel head from larry cars running on tracks located above the chambers. The coal is leveled by means of a mechanical leveler, and combustion is maintained by the admission of air from both ends of the chamber; after the process is finished, the coke is watered, and the resultant product is then pushed out of the chamber by means of a mechanical pusher which delivers the coke to a conveyor, this conveyor being usually so arranged as to deposit the coke into cars. It is claimed that this oven gives an increased yield, better coke, and a lower operating cost than does the usual bee-hive oven.

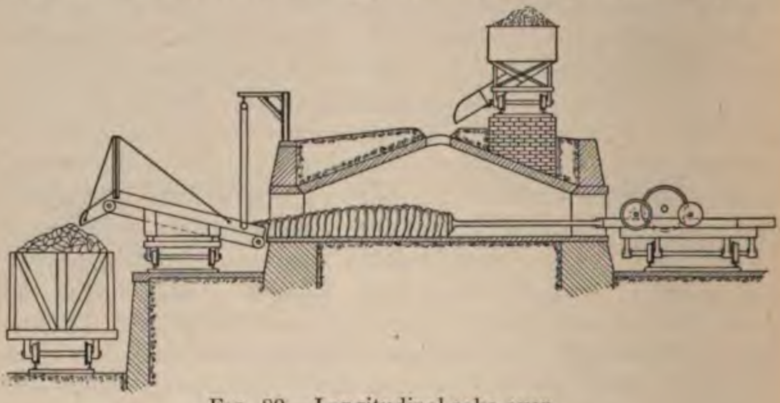

Fig. 80.—Longitudinal coke oven.

The draft[1] in both the bee-hive and the longitudinal oven should be regulated in such manner as to secure the maximum degree of heat produced through the burning of the volatile matter of the coal at a point located above the charge, thus preventing the burning of the fixed carbon of the coal, and in this manner leaving the oven in the proper state of preparedness to receive the next charge of coal, and the volume of air admitted should be decreased in the same ratio as the decrease in volatile matter expelled and burned. BELDEN states that the study of this feature of operation shows that proper attention, especially toward the latter part of the coking process, is not given and that failure to close off the draft at the proper time delays coking at any stage, and during the last few hours of the coking process this lack of attention results in burning up the coke, in cooling the oven, and thus placing the latter in a bad state of preparedness to receive a subsequent charge. Thus inattention requires that the subsequent charge shall be considerably reduced in weight, in order that proper operating conditions may be again secured; or that "black butts" are produced, due to the fact that the oven, which has thus been materially delayed in catching up, does not come off at the proper time, and the charge is not properly coked at the bottom.

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[1] Belden, Metallurgical Coke, Tech. Paper 50, Bureau of Mines.

The term "black butts" designates the soft black ends of coke found at the bottom of the prisms, and they are usually produced by a cold or dirty bottom, as well as by lack of proper draft regulation during the latter period of the coking process, this latter condition also causing a slowing down of the process by a decrease in temperature, thus leaving volatile matter in the coke. BELDEN further states that "the use of too large charges, usually in an endeavor to increase the output during prosperous times, always causes black butts, and is poor policy, generally defeating the very object for which it is tried. An oven of a given size will, under the most favorable conditions, produce only a certain amount of coke from a given coal, and although the weight of merchantable product drawn may be slightly increased for one or two charges, the reduction of the amount of coal charged subsequently to get the oven back to condition, and the amount of breeze carted away, will show on the wrong side of the ledger every time. It is much better to find out the maximum amount of coal that can in a given time produce the maximum amount of good coke and the minimum amount of breeze and ashes, and then to insist that the output be kept at this figure."

By-product Ovens.—The introduction of the by-product oven has caused, and is still causing quite a change in the economic geography of the coal fields producing coal available for coking purposes; bee-hive ovens are usually located in close proximity to the mines which supply the coal for coking and when, as is natural, these mines become exhausted, it becomes necessary to abandon the bee-hive plant. Again, many coals are not suitable for coking in bee-hive ovens, and the attempt to secure by-products from such plants invariably ends in failure.

The by-product oven plant can readily be constructed in the immediate vicinity of the blast furnaces using the coke, because it is entirely practicable to ship coking coals to the plant from any point which may lie within a favorable freight rate zone. Besides this, those coals which are not suitable for bee-hive practice, can readily be used, when properly mixed, in the by-product oven, thus making more coals available for coking and, at the same time, producing an acceptable blast-furnace coke. By-product ovens permit of the entire recovery of the valuable gas constituents which are sent to waste in the bee-hive practice, while the cost of production in by-product ovens, when located at the blast-furnace, is considerably less than is the cost of bee-hive coking at the mines with consequent transportation to the furnaces, this being especially true when the bee-hive ovens are located a considerable distance from the furnaces. Even though the original investment in by-product ovens, and its recovery plant, seems at first sight to be excessive, the returns secured from the resultant by-products are such that the profits readily permit of this additional outlay.

C. A. MEISSNER,[1] in referring to the relative costs of bee-hive and by-product ovens, says:

> "A great deal has been said about the comparative cost of by-product coke ovens versus bee-hive ovens. We (United States Steel Corporation) have studied this very carefully and find that this is dependent entirely on the location of the bee-hive oven plant and the attendant conditions of such a location. Comparing, for instance, a by-product plant at Gary, Ind., with a bee-hive plant in the Pocahontas, West Virginia, region, both on a large scale and taking into consideration the mine investment, houses required for workmen, and everything connected with the construction ready for operation of either type, we find that the bee-hive oven installation complete costs more than the by-product oven installation complete for the same number of tons of coke produced per year. Comparing the above by-product plant with a similar bee-hive oven installation in the Connellsville region, where all conditions are more favorable and where the difference in the yield of coke from coal coked in the bee-hive oven and in the by-product oven is materially less than it is in the Pocahontas region, we find that the cost of a bee-hive oven installation complete is considerably less than

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[1] U. S. Steel Corporation, Paper on "The Modern By-product Coke Oven," 1913.

the cost of a by-product coke oven installation complete, per ton of coke produced per year.

"A feature of supreme importance in the use of the by-product oven is the greater coke yield obtained in this type than in the bee-hive oven from the same coals. This amounts to about 23 per cent. to 25 per cent. in the low volatile coals, such as POCAHONTAS and from 5 per cent. to 12 per cent. in the high volatile coals, dependent on their original content of volatile matter. This subject is not fully understood or generally appreciated and yet is of vast importance in the conservation of our coal fields.

"A concrete example will be of interest:

	Pocahontas coal coked in	
	Bee-hive ovens	By-product ovens
Number of ovens	6,154	560
Coking time	72 hours	17.5 hours
Yield of coal to coke, per cent	60	82
Net tons of coal required to produce 2,880,000 tons of coke per year	4,800,000	3,512,000
Net tons of coal saved per year by use of by-product coke ovens for above coke production		1,288,000

"This amount of coal saved, if it were coked in by-product ovens, would produce about 1,000,000 tons of coke per year. What this means in coal conservation is so plain that no further comment is necessary."

The present systems of by-product oven construction are all of two types, the one being based on the horizontal flue construction of SIMON-CARVES, and the other on the vertical flue of COPPÉE; no matter which of these two systems is followed, it will be found that the coking chamber proper usually consists of a long, narrow, retort-shaped structure, built up of firebrick in such manner that a series of these retorts constructed side by side form a "battery" of ovens.

The dimensions of the retorts, as well as of the ovens generally, are not alike, various constructors varying these dimensions to suit their individual system; but, generally speaking, the ovens or retorts are about 33 ft. long, from 17 to 22 in. wide, and about 6½ ft. high, the width of the retort tapering toward the discharge end in such manner that the width at this end is from 1 in. to 4 in. more than at the other end; the amount of this taper is dependent upon the nature of the coal to be coked, varying from 1 in. for coals that shrink, to 4 in. for coals that swell.

MR. C. A. MEISSNER, in referring to the width of the retort, or oven, states that in his opinion the present width of oven, averaging about 19 in., may not be the limit to which they may be built and still produce a satisfactory coke within a reasonable coking time. Modern oven construction permits of such a flexible heating arrangement that, in his opinion, with a wider oven sufficient increased heating can be so applied as to attain a coking time similar to that in a narrower oven. MEISSNER further states

> "As a matter of fact, we built our early ovens for a 24-hour coking time after considerable discussion on this very subject of width of oven in connection with coking time. On the same width of oven we reduced this coking time down to 16.5 or 17 hours. At the Birmingham, Ala., ovens the taper was decreased, causing an average increase in width of ¾ in. more than in the Gary, Ind., ovens. This was for a high volatile coal mixture compared to the Gary, Ind., mixture, and yet the coking time at Birmingham is down to 17.2 hours and will undoubtedly be still further reduced. The coke ovens at Farrell, Pa., with a width of 17 in., and no taper, have a coking time of 20 hours. The Riverside coke ovens, with a 17-in. width, are running on a 20-hour coking time."

The retort, or oven ends are closed by means of firebrick-lined iron doors, the doors being made to fit as tightly as possible and then sealed, or luted with clay. This procedure is very expensive as regards both labor and material, and attempts are being made to provide a self-sealing door, similar to those used on the retorts of gas benches, for this purpose. If the door is not properly sealed it will lead to leaks, aggravated by pressure variations, and thus burn up a portion of the coke besides causing a deterioration in the value of the by-products, as well as in the resultant gas.

Coal is charged into the tops of the ovens through trunnel heads, while the gas evolved during carbonization is led away by means of ascension pipes leading to a common collecting main placed on top of the battery of ovens. The coal in the ovens is leveled and pushed by means of mechanical devices, and the coke is watered, or quenched after being pushed from the retort.

In the early days of by-product coking the expansion produced in a battery of ovens often led to serious trouble, but the extent of this expansion in any direction is now pretty well understood by the builders, and proper precautions are taken to avoid these difficulties. The number of ovens in a battery is often a choice of operating conditions, varying from 40 to 100, while Germany exhibits one battery with as many as 120 ovens; there should be no practical limit to the number of ovens in a battery, providing proper precautions have been taken to care for the expansion occurring between individual ovens.

Coking in by-product ovens is a true distillation process, and its regulation is under scientific control. The heat required for distillation is supplied by burning a portion of the gas, evolved from the coal, in flues which surround the oven, the air required for combustion being furnished from the outside; and this with or without regeneration, the method of preheating the air, or the adoption of regenerators, being determined by the demand, if any, for surplus gas to be used for other purposes; if regenerators are used, the surplus gas usually amounts to from 40 to 60 per cent. of the total gas evolved from the coal.

BELDEN[1] states that in all systems, the principal object to be achieved is a uniform heating of the oven, and the builders of the various systems, whether with vertical or horizontal flues, all claim to obtain this uniform heating by means of their own particular methods of regulating the burning of the gas. Uniform heating results in a uniform coke throughout the entire charge, and it also reduces the period of carbonization; on the contrary, uneven heating of the oven causes a more rapid coking of the coal at the points of higher heat, thus making a non-uniform product, besides increasing the time of carbonization and thus reducing the maximum output per oven per unit of time, even though it may not reduce the actual yield of coke from the coal charged.

In the by-product oven the coking process proceeds from the sides of the oven to the middle, and a marked line of cleavage, Fig. 81,[1] should appear down the center of the entire charge, the coke produced having a short, blocky structure, as distinguished from the long columnar structure of coke produced in the bee-hive oven, Fig. 82,[1] in which coking proceeds from the top downward.

BELDEN further states that

> "Since no fixed carbon is burned during the process of by-product coking, and since in this process conditions are more favorable for the breaking up of the heavier hydrocarbons, with subsequent deposition of carbon on and through the coke, the yield from any coal in by-product ovens is higher than the yield from the same coal in bee-hive ovens. The percentage of impurities is reduced in proportion to the increased yield. Statistics collected by the U. S. Geological Survey give the average yield of by-product ovens as 75.1 per cent., compared with 64.7 per cent. for bee-hive ovens."

The by-product yield is usually due to operating conditions, and it is possible for the operator to sacrifice the quantity of his coke in order to secure a better quality as well as an increased yield of by-products and of gas, or conversely, he can sacrifice the latter for the former; in this manner the operation of the ovens depends upon which product is the more desired. Ideal operating conditions are those which permit of producing a coke which shall possess all of the properties required in a coke for a specific purpose together with a maximum yield and quality of gas and by-products. The treatment of the gas evolved from the coal is dependent upon the manner in which it is to be used; if the surplus gas is to be used for fuel purposes only, the battery is usually supplied with only one gas main, but if the surplus gas is also to be used for illuminating purposes, the battery must be supplied with two gas mains—the one for the collection of the rich gas evolved during the early period of carbonization, and the other for collecting the poor, or fuel gas evolved during the later period. Here again we find quite some difference in the method of operation, as the rich gas collected will depend entirely upon the illuminating power required, the illuminating standard being highest during the first hours of the carbonizing period. Usually, however, the first half, or thereabouts, of the gas evolved is used for illuminating purposes, or the gas evolved during the first 6 to 10 hours of the carbonizing period, this time of course being again dependent on the length of time required for complete carbonization. The two gases thus separated during production are kept separate during the entire subsequent operations; therefore two sets of condensing and by-product apparatus, usually placed in parallel, are required for treating the gas.

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[1] Tech. Paper 50, Bureau of Mines Metallurgical Coke, A. W. BELDEN.

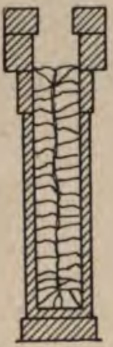

FIG. 81.—Structure of by-product oven coke.

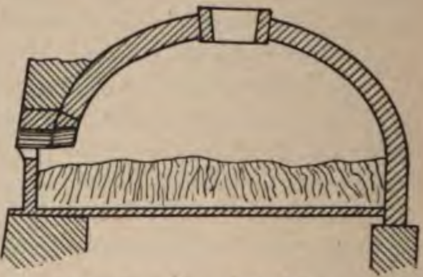

FIG. 82.—Structure of bee-hive oven coke.

If the gas evolved is to be used for illuminating purposes, it is generally enriched by the addition of benzol vapors after its proper condensation and purification, and is then led to the gas-holder, ready for use, while the poor, or fuel gas, after its by-products have been removed, is stored for heating the carbonizing chambers and for other fuel purposes. The gas in leaving the ovens passes up through the ascension pipes into the collecting main and is then passed through the coolers, scrubbers, exhausters, tar extractors and ammonia washers to the gas holder. Unless sulphate of ammonia is to be produced by the direct or the semi-direct systems,[1] the ammoniacal liquor coming from the ammonia washers is distilled, the ammonia being then passed into a sulphuric acid bath for the production of sulphate; or it may be condensed for the production of strong ammonia liquor, varying in strength from 15 to 20 per cent. If it is required to enrich the illuminating gas, the benzol is removed from the poor gas before it is piped to the storage holder, and this benzol is used for enriching the rich gas.

Fig. 83 is a schematic plan showing the arrangement of a gas-treating plant in which neither the direct or the semi-direct sulphate process is used. Continuous operation of a by-product coke-oven plant usually results in producing the same quantity of surplus gas daily, and if this product is up to the minimum daily consumption, the fluctuation between minimum and maximum output can be taken care of by either installing a producer plant, or a water-gas plant. If producers are used, they can be operated on coke screenings or refuse coke, and the gas thus obtained can be mixed with the poor coke-oven gas and used as fuel under the ovens, thus permitting of drawing off more rich gas, some of which however, would have to be enriched to maintain the illuminating standard.

In the condensing plant illustrated in Fig. 83, we find the poor gas and the rich gas entering separate apparatus, first passing through the air and water coolers, then through the tar extractor, exhauster, secondary cooler, and ammonia washers, the secondary cooler being required to reduce the temperature of the gas occasioned by compression in the exhausters. Up to this point both gases are treated alike; the rich gas now leaves the condensing plant and enters the purifiers, passing from thence into the storage holder which feeds the city mains. The poor gas, after leaving the ammonia washers, is passed through the benzol washers (*A*) and (*B*), and passes into the storage holder (*C*). Here, if necessary, it is mixed with producer gas, as explained above, and then passes on to the ovens, while the benzol removed from the poor gas is used for enriching the rich gas and thus increasing its candle

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[1] See "Coal Gas Residuals," McGraw-Hill Book Company, Inc., 1914.

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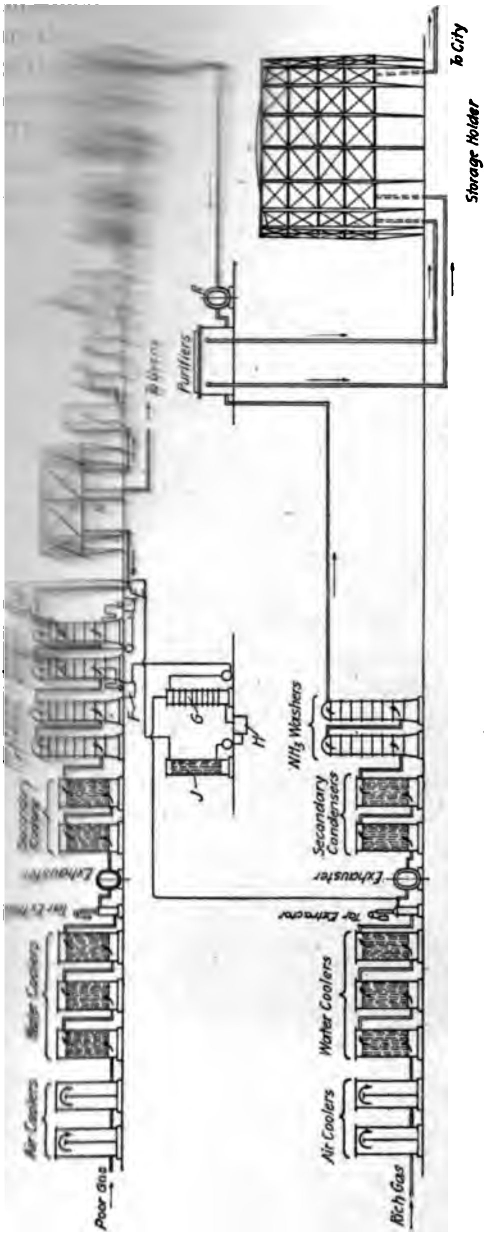

Fig. 83.—Coke-oven gas-condensing plant.

power. The tar oil used for removing the benzol from the poor gas is run from tank (*D*) through the second benzol washer (*B*), and thence into tank (*E*), from whence it is pumped into the first benzol washer (*A*). The tar oil coming from tank (*D*) contains about 5 per cent. of benzol, but when it finally leaves washer (*A*) its benzol content has been increased to about 15 per cent., and it is then collected in tank (*F*). From tank (*F*) this benzol laden oil is pumped into the still (*G*), where the benzol content is again reduced to about 5 per cent., the exhausted oil from the still being collected in tank (*H*), from whence it is pumped through the cooler (*J*), and is then led to tank (*D*) again, being ready for use in the washer (*B*). Sometimes a distillation plant, consisting of a simple tar still and condenser is added to the system, for the purpose of revivifying a small portion of the oil, in order to maintain its benzol extracting properties. In this still the oil is freed from the greater portion of the naphthalene and other hydrocarbons absorbed during the process; the presence of an excessive percentage of these constituents in the tar oil would render it inefficient as a benzol extractor. The benzol vapors evolved in the still (*G*) are now carried into the rich gas system by means of a gas current coming from the pressure side of the rich gas condensing plant, the mixture of gas and benzol vapors then passing to the inlet of the exhausters of the rich gas plant. Operating in this manner precludes the possibility of the vapors coming into contact with a large body of tar, as the latter would absorb a considerable quantity of the benzol.

If the auxiliary water-gas plant is made a part of the system, its operation is subject to three methods, viz.: either as a producer plant, as a carburetted water-gas plant, or as a combined producer and water-gas plant. If the plant is to be used simply as a producer, the valve (*a*) may be kept permanently closed, while valve (*b*) is opened. If air and steam are now blown through the incandescent fuel bed in the producer (*K*), the resultant gas, after being scrubbed in the scrubber (*L*) may be passed into the fuel gas holder, and there mixed with the poor coke-oven gas.

If the apparatus is to be used as a carburetted water-gas plant, valve (*b*) is closed, and valve (*a*) is opened, the ordinary system of generator (*K*), carburetor (*M*), superheater (*N*), wash-box (*O*), condensers (*P*), and relief holder (*Q*), being followed. This carburetted water gas is now taken from the holder (*Q*) by the exhauster (*R*) and sent into the rich gas purifying plant, where it mixes with the latter and then passes on into the city storage holder.

If the system is to be used as a combined producer and water-gas plant, valve (*b*) should be opened while blowing air through the generator (*K*) and the producer gas will then pass through the scrubber (*L*) into the fuel-gas holder (*C*). When the fuel bed has thus been made incandescent, valve (*b*) is closed and valve (*a*) is opened, steam being then blown through the generator, and the resultant blue water gas is passed through the entire system, but no oil is added in the carburetor; this blue water gas is now piped to the city storage holder, where it is mixed with the rich coke-oven gas, the lack of illuminating power in the blue gas being made up by additional benzol vapors in the rich coke-oven gas.

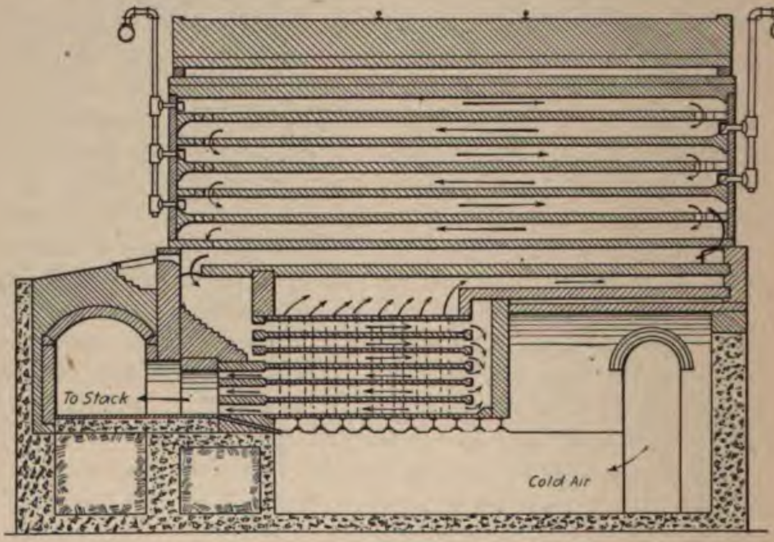

FIG. 84.—Semet-Solvay recuperative oven.

This system is, of course, only one of many and was practised by the late DR. SCHNIEWIND of the United Coke and Gas Company; each individual oven builder advocates his own method of gas treatment, but the essential feature of all will be found similar to the above.

Semet-Solvay Recuperative Oven.—Until very recently this oven was only of the recuperative style, the regenerative system not being applied except in later constructions, and the heating of the retort was accomplished by means of horizontal flues instead of vertical ones, Fig. 84. The modern oven in the United States is the result of a gradual evolution of the old three-flue oven, and it is now generally constructed with six flues. The necessary gas for combustion is supplied to each horizontal flue by means of a burner pipe which enters the end of the flue, alternating at opposite sides, while the air required to support combustion is drawn by stack draft through the recuperators located in the base of the oven, passing from there into risers located at the ends of the oven, and then through an opening at the end of the flue into the horizontal combustion flues, the requisite air being regulated by a brick damper in each flue. Combustion starts in the top of the flue, the resultant products traversing each horizontal flue, while the heat required to maintain a uniform temperature in the setting is secured by admitting fresh air and flue gas to each horizontal flue in proportion to its requirements. The flue is given a gradual increase in size from the top to the bottom in order to allow for the greater volume of combustion products evolved, but the bottom flue has a decreased cross-section, because only a portion of the combustion products is sent through the same. The combustion products finally pass out into the recuperators, a portion coming from the bottom flue and another portion from a sole flue located beneath the oven, the proportion from each flue being regulated by means of brick dampers. The SEMET-SOLVAY oven is usually provided with division walls 18 in. thick, thus permitting the operation of any one oven while the neighboring one is down for repairs.

Recent ovens of the SEMET-SOLVAY type have been built on the regenerative, instead of the recuperative principle, and the design of this added feature is very simple, each oven being provided with two regenerators, located below the oven, and supplied with chimney flues between them. Operating conditions require that the reversal of the gases shall take place throughout the entire system of horizontal flues and regenerators, thus causing the current of gas to flow upward through the flues during one period, and downward during the succeeding period; this reversal of flow is automatically governed by the gas currents without any attention from the operator, and the heating gas is supplied in a continuous stream. The regenerative ovens are especially adapted to such coking conditions as require a maximum temperature, and also to such installations where it is desirable to secure a maximum amount of surplus gas, but little attempt being made to utilize the waste heat for steam making; in some cases, however, the introduction of waste-heat boilers, located between the regenerators and the stack, may be economically applied, due to the fact that the air has capacity for absorbing only a portion of the heat evolved from the products of combustion, but such a case must be worked out on its own merits. This type of oven is illustrated in Fig. 85, and it may be stated that selection of either type, whether recuperative or regenerative, will depend principally upon both commercial and operating conditions as applied to each individual plant.

Semet-Solvay Ovens at Detroit.—The SEMET-SOLVAY ovens at Detroit, Mich., first delivered their surplus gas to the Detroit City Gas Company in 1902, the ovens being 30 ft. long, 7 ft. high, and having an average width of 16.5 in., each regular charge amounting to 6 tons of coal. Each oven is provided with four flues, one above the other, the flue walls being constructed of large Belgian tile, while the division walls and recuperator flues, the latter being located directly below the oven proper, were built up of clay firebrick. After leaving the recuperators, the products of combustion were piped to waste-heat boilers for the production of steam and, after passing through these boilers, they escaped to the atmosphere at an average temperature of 550°F. (288°C.).

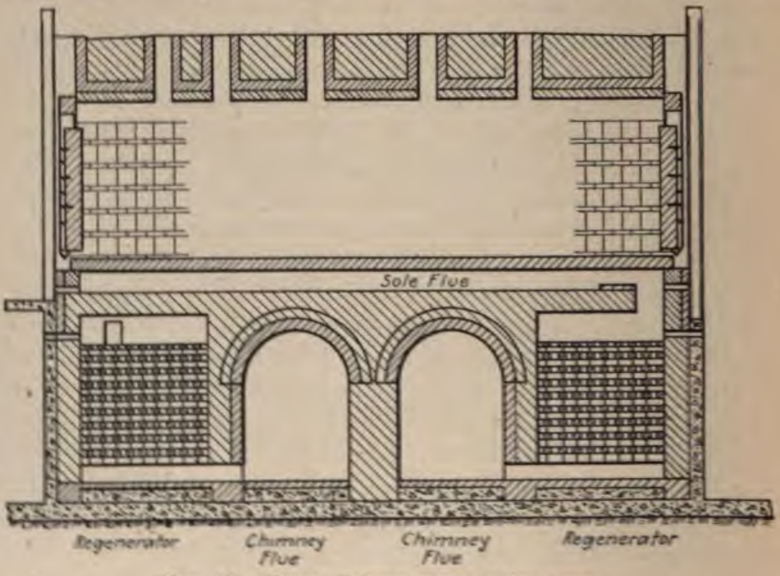

FIG. 85.—Semet-Solvay regenerative oven.

In a more recent installation of these ovens,[1] the carbonizing chambers were made 35 ft. long, and the heating flues were increased to five, silica brick being used in the division walls as well as in the recuperator flues. The air required for combustion in these new settings is heated to about 1400°F. (760°C.), or about 600°F. (315°C.) higher than in the old type of recuperator. A better heat transfer from the combustion gases to the air was secured in the new ovens, and this resulted in a decreased fuel consumption in the producer, but, as the gases leave the recuperators with a temperature lower by about 500°F. (260°C.) than in the original type, they have a much reduced value for steam generating purposes.

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[1] W. S. BLAUVELT, A. G. I., 1911, page 481.

Quite some pains were taken to determine the type of oven best suited for the greatly varying local conditions; in consequence 12 different types of oven were built, these ovens being rebuilt from time to time, particular attention being paid to the heating system, but in all of them the air was preheated in sole-flue recuperators. The results obtained with the various types of SEMET-SOLVAY ovens tried out at Detroit are given below:

Type number	1	2	3	4	....
Number of flues high	4	5	5	5	5
Division wall material	Fireclay	Fireclay	Silica	Silica	Silica
Fuel gas required, cu. ft.	6800	6200	5500	5300	....
Fuel gas, B.t.u. per cubic foot	525	517	505	501	....
Surplus gas, cu. ft.	3900	4500	5200	5400	10,700
Surplus gas, B.t.u. per cubic foot	610	610	610	610	556
Pounds of steam from waste heat	950	900	800	400	500
Pounds of coke required for oven fuel	....	....	....	....	235

In the original installation each block of ovens was equipped with a double hydraulic main, this main being provided with separate compartments for rich and poor gas, and duplicate condensing and washing systems were also provided for both kinds of gas.

The gas, as originally delivered, had a nitrogen content varying from 3 to 12 per cent., while the calorific power of the gas varied as much as 75 B.t.u. per cubic foot, this condition being due to leakage through the oven walls, the leakage being induced by variations in pressure. Experiments showed that a satisfactory yield of gas, with a uniform nitrogen content below 4 per cent., could be obtained from a block of ovens only when the variation in gas pressure in the hydraulic main was less than 0.03 in. of water, and it became necessary to develop a governor which was capable of controlling the pressure variations within this limit.

The coal used gave a gas of about 700 B.t.u. per cubic foot when the oven was freshly charged, this heating value being reduced to 350 B.t.u. at the conclusion of the coking period, and if all of the surplus gas were from the richest portion of the product, it would average about 650 B.t.u. The connections are so arranged that the gas from any oven may be delivered to the rich or poor gas system at will, and, therefore, the separation of gases can be so varied as to deliver all of the surplus gas at any calorific value between 500 and 640 B.t.u. In practice it became customary to vary the calorific power with the illuminating power as follows:

	B.t.u.
15-candle-power gas	605
16-candle-power gas	620
17-candle-power gas	635
18-candle-power gas	640

In order to meet the requirements of the Detroit City Gas Company, a water-gas set was installed for enriching the surplus gas, but since March, 1904, benzol was used for this purpose. The gas is now washed before enrichment with an absorbing benzolized oil which practically removes all the naphthalene and other condensible hydrocarbons, and absorbs but little benzene. But little difference has been found between the candle powers immediately after enrichment and those observed by the Gas Company at a point 2.5 miles distant, the enriching value of the benzol being as high in the winter months as during the summer.

One of the principal objects of this plant was to secure an exceptionally high grade of foundry coke, and, in order to secure this result, it became necessary to use a mixture of coals, one of the coals being very low in volatile matter. The best foundry coke was secured from a coal mixture which contained from 25 to 26 per cent. of volatile matter, but such a coal yielded a lower gas volume, and one which was also lower in volatile products than would have been obtained from a good gas coal containing from 32 to 34 per cent. of volatile. In spite of the necessity of using the low volatile coal, the normal yield of gas per ton of dry coal, the nitrogen content in the gas not exceeding 4 per cent., varies between 10,400 and 10,800 cu. ft. when coking in 20 hours. With higher heats and shorter coking periods, the gas yield can be increased to 11,500 cu. ft.

Semet-Solvay Ovens at Waukegan.—This plant consists of 13 ovens of the SEMET-SOLVAY type, the ovens being 30 ft. long with an average width of 18¼ in., and taper of 1½ in., each oven being five flues high and receiving a charge of 8.5 tons of coal, Those portions of the ovens which are subjected to high heats are built entirely of silica material, the ovens being fired with individual producers located below and in front of each oven, using small coke or coal as fuel. The waste gases from the combustion flues are drawn by steam-driven SIROCCO fans through two boilers of 240 hp. each, these boilers furnishing all of the steam required for the plant. The coal used is straight Youghiogheny and the coke produced, which is of a satisfactory quality, is sold mainly for domestic purposes, although, by the use of coal of proper quality, it is expected that the plant can turn out a high-grade foundry coke. The ovens are so designed that, operating under heats which produce the best quality of coke, the charges can be carbonized in 24 hours. The fuel consumption in the producers averages between 250 and 260 lb. of dry coke per ton of coal carbonized, and it is expected that this figure will be improved. The gas produced amounts to 10,760 cu. ft. per net ton of coal with an average of 576 B.t.u. per cubic foot, while the by-products show a little more than 23 lb. of sulphate and 10 gal. of tar per net ton of coal carbonized.

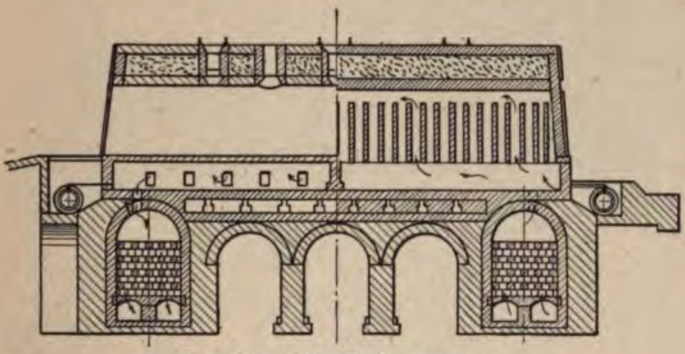


FIG. 86.—Otto-Hoffman oven.

The Schniewind Type of Otto-Hoffman Oven.—The OTTO-HOFFMAN oven is shown in Fig. 86, while the SCHNIEWIND modification of this oven is shown in Figs. 87 and 88, the latter oven (1) being an air-tight retort consisting of a rectangular chamber 43 ft. 6 in. long, 17 in. wide, and 6 ft. 6 in. high. The ovens are placed side by side, and are supported on a steel structure running the entire length of the battery, this construction permitting of an inspection of the brick-work at all points. The retorts are separated by hollow walls divided into ten compartments (2), each compartment containing four vertical flues (3) while an air chamber (4) is located directly beneath the carbonizing chamber. Alongside this chamber, and directly under the vertical flues above referred to, are ten combustion chambers (5), the gas supply to each of these chambers being supplied with an independent control, thus permitting of a uniform heat throughout the entire length of the oven. The air for combustion is admitted through the opening (6) in the wall between the air and the combustion chambers, this air being heated to 1800°F. (980°C.) by a pair of regenerators (7) placed under the center of the battery and running along its entire length. A vertical flue (8) conducts the air from the regenerator to the air chamber (4) under the oven. The SIEMENS regenerative principle is employed, the air being reversed twice each hour, while the fuel gas is simultaneously reversed by a suitable valve, but the gas is not regenerated. The gas unites with the preheated air in the combustion chambers (5), ascends through the vertical flues (3) to a horizontal flue (9), through which it passes and then descends through the five chambers in the other end of the oven, after which it passes through the air chamber

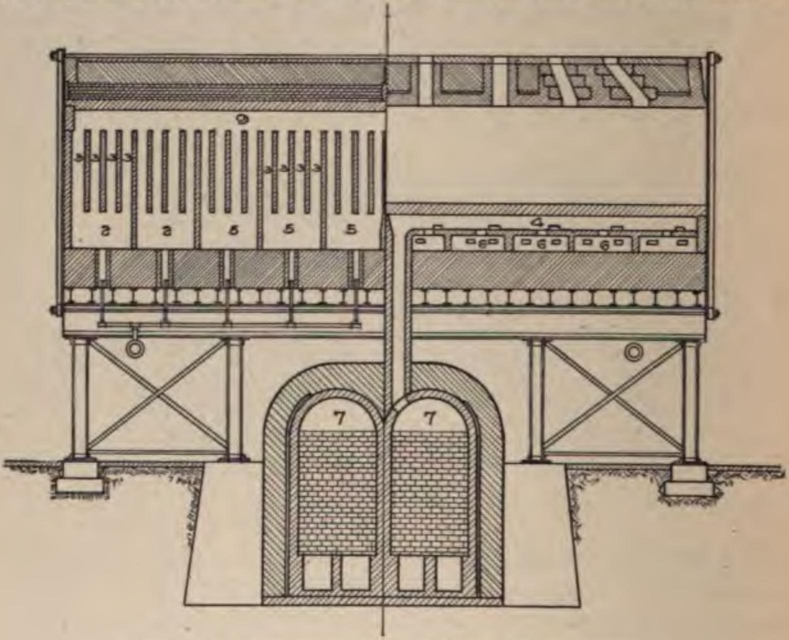


FIG. 87.—Schniewind type of Otto-Hoffman oven.

(4) and vertical flue connection (8) to the regenerator (7), and from thence through the reversing valves to the chimney. The regenerators are built entirely independent of the oven structure, so that their expansion does not affect the brick-work of the oven.

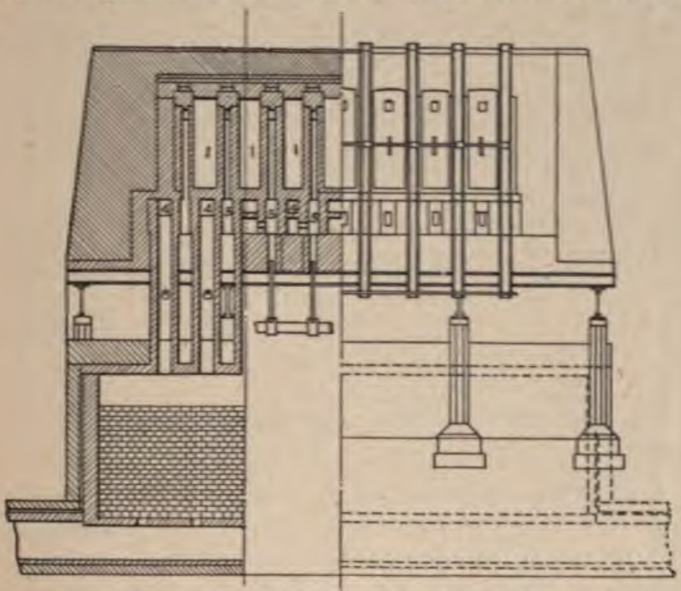


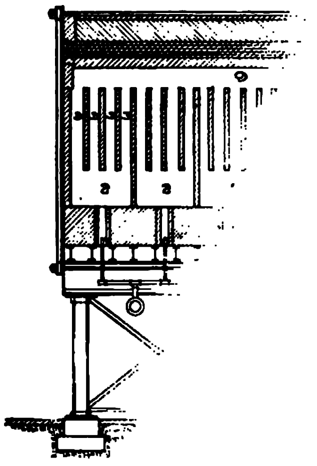
FIG. 88.—Schniewind type of Otto-Hoffman oven.

Table LXXV gives the results obtained with various coals coked in OTTO-HOFFMAN ovens, and although all of the coals used were true bituminous coals, the results show wide variations.

TABLE LXXV.—YIELDS OF VARIOUS COALS USED IN OTTO-HOFFMAN OVENS

Plant used	Coke, per cent.	Tar, per cent.	Sulphate, per cent.	Total gas per 2000 lb., cu. ft.
Average operating results with Dominion				Approx.
(Cape Breton) coal at Everett, Mass.	72.83	4.99	1.010	9,000
Youghiogheny (Pa.) coal at Glassport, Pa.	75.60	5.07	1.100	9,000
Westphalian coal, Germany	74.50	3.70	1.280	9,600
Distillation test:				
Connellsville, Pa., coal	76.34	6.14	1.223	8,924
Pittsburgh, Pa., coal	68.25	4.38	0.908	8,884
Coal from eastern part of Pennsylvania	85.00	2.00	0.800	8,400
Virginia coal	66.01	4.70	1.070	10,090
Kanawha, W. Va., coal	73.60	6.40	1.000	10,289

rated by hollow walls divided into compartment containing four vertic ber (4) is located directly ben Alongside this chamber, and above referred to, are ten comb to each of these chambers be control, thus permitting of a length of the oven. The ai the opening (6) in the wal



he International Breton, Nova

	Pounds	Per cent.
	1593.4	71.13
	75.7	3.38
	7.6	0.34
	368.0	16.43
	10.8	0.48
	1.6	0.07
	182.9	8.17
	2240.0	100.00

e coal used, and the coke produced,

	Coal, per cent.	Coke, per cent.
	75.10	86.42
	3.75	1.06
	1.51	0.73
	11.05	0.46
	5.84	8.91
	2.75	2.42
	100.00	100.00

che determined by the DULONG-MAHLER for- re B.t.u. for the coal and 13,305 B.t.u. for substance, free of ash, contained 2.52 per hydrogen, while the specific gravity of the dry

of the coke ash is of great importance, espe- to be used for domestic purposes, because the is liable to form undesirable clinkers, DR. the International Dominion ash and com- ash from other cokes and coals as follows:

LXXVI.—ANALYSES OF ASH

		[illegible] coal	Connellsville coke by E. PECHIN	Phelan coke	Anthracite from Lehigh Coal & Navigation Co.'s Panther Creek mine	
					Red ash	White ash
[illegible]	SiO_2	27.71	57.48	26.65	47.19	48.25
[illegible]	Al_2O_3	13.04	34.64	12.50	32.52	36.18
[illegible]	Fe_2O_3	50.60	5.09	46.45	4.71	3.29
[illegible]	Mn_3O_4	0.25		0.70	Trace	Trace
	CaO	4.61	2.58	6.15	3.64	1.95
	MgO	0.77	0.08	1.43	0.97	0.92
[illegible]	K_2O	0.85	Trace	0.64	7.31	7.25
[illegible]	Na_2O	0.18		0.41		
[illegible] acid	SO_3	2.62		5.01	0.71	0.49
[illegible] acid	P_2O_5	0.10	0.13	0.25	1.96	0.92
[illegible] acid					0.99	0.75
Total		100.00	100.00	100.00	100.00	100.00

The tar yield in the above test amounted to 3.38 per cent., but DR. SCHNIEWIND states that the actual yield at Everett, Mass., amounts to 4.99 per cent., explaining the difference in yield to the fact that the coal used in the test had been exposed to the weather for some time. Table LXXVII shows the results secured during fractional distillation of the International Dominion tar, as well as a German tar, the latter for comparison.

TABLE LXXVII.—TAR DISTILLATION PRODUCTS

Fractions	Temperature, °C.	Otto-Hoffman coke oven tar			
		Dominion coal test	Dominion coal at Everett	Dominion coal at Sydney	German plant at Westphalia
Light oil	90–170	3.7	1.26	1.34	[illegible]
Middle oil	170–230	9.4	14.74	11.46	10.54
Heavy oil	230–270	12.0	7.07	8.56	7.62
Anthracene oil	Over 270	43.0	21.38	20.62	44.25
Pitch......		27.0	53.03	53.68	30.55
Water.....		2.4	1.32	1.93	Trace
Loss.....		[illegible]	1.01	1.38	[illegible]
Total		100.0	100.00	100.00	100.00
Fixed carbon, per cent.		[illegible]	[illegible]	[illegible]	
Specific gravity		[illegible]	[illegible]	1.14	[illegible]

A summary of results secured with coal from the International Seam of the Dominion Coal Company at Cape Breton, Nova Scotia, gives:

Product of 1 long ton of coal	Pounds	Per cent.
Coke	1593.4	71.13
Tar	75.7	3.38
Ammonia (=1.373 per cent. sulphate)	7.6	0.34
Gas, total 10,390 cu. ft. of 0.466 sp. gr.	368.0	16.43
Sulphur compounds in gas:		
H_2S = 0.98 lb. per M cu. ft.	10.8	0.48
CS_2 = 0.13 lb. per M cu. ft.	1.6	0.07
Gas liquor and loss, by difference	182.9	8.17
Total	2240.0	100.00

The ultimate analyses of the coal used, and the coke produced, showed:

	Coal, per cent.	Coke, per cent.
Carbon	75.10	86.42
Hydrogen	3.75	1.06
Nitrogen	1.51	0.73
Oxygen	11.05	0.46
Ash	5.84	8.91
Sulphur	2.75	2.42
Total	100.00	100.00

The calorific value, determined by the **Dulong-Mahler** formula, gave 12,437 B.t.u. for the coal and 13,305 B.t.u. for the coke. The coal substance, free of ash, contained 2.52 per cent. of disposable hydrogen, while the specific gravity of the dry coal was 1.28.

As the composition of the coke ash is of great importance, especially if the coke is to be used for domestic purposes, because the more fluxible ash is liable to form undesirable clinkers, **Dr. Schniewind** analyzed the International Dominion ash and compared it with the ash from other cokes and coals as follows:

TABLE LXXVI.—ANALYSES OF ASH

Constituent	Symbol	International coke	Connellsville coke by E. PECHIN	Phelan coke	Anthracite from Lehigh Coal & Navigation Co.'s Panther Creek mine	
					Red ash	White ash
Silica	SiO_2	27.71	57.48	26.65	47.19	48.25
Alumina	Al_2O_3	13.04	34.64	12.50	32.52	36.18
Oxide of iron	Fe_2O_3	50.60	5.09	46.45	4.71	3.29
Oxide of manganese	Mn_3O_4	0.25		0.70	Trace	Trace
Lime	CaO	4.61	2.58	6.15	3.64	1.95
Magnesia	MgO	0.77	0.08	1.43	0.97	0.92
Potassium oxide	K_2O	0.85	Trace	0.64	7.31	7.25
Sodium oxide	Na_2O	0.18		0.41		
Sulphuric acid	SO_3	2.62		5.01	0.71	0.49
Phosphoric acid	P_2O_5	0.10	0.13	0.25	1.96	0.92
Titanic acid					0.99	0.75
Total		100.00	100.00	100.00	100.00	100.00

The tar yield in the above test amounted to 3.38 per cent., but DR. SCHNIEWIND states that the actual yield at Everett, Mass., amounts to 4.99 per cent., explaining the difference in yield to the fact that the coal used in the test had been exposed to the weather for some time. Table LXXVII shows the results secured during fractional distillation of the International Dominion tar, as well as a German tar, the latter for comparison.

TABLE LXXVII.—TAR DISTILLATION PRODUCTS

Fractions	Temperature, °C.	OTTO-HOFFMAN coke oven tar			
		Dominion coal test	Dominion coal at Everett	Dominion coal at Sydney	German plant at Westphalia
Light oil	80–170	3.7	1.26	1.38	6.55
Middle oil	170–230	9.8	14.73	11.46	10.54
Heavy oil	230–270	12.0	7.07	8.56	7.62
Anthracene oil	Over 270	43.0	21.38	20.63	44.35
Pitch		67.0	53.03	53.68	30.55
Water		2.3	1.52	1.93	Trace
Loss		0.9	1.01	1.38	0.39
Total		100.0	100.0	100.0	100.00
Fixed carbon, per cent.		8.0	8–10	5.35	
Specific gravity		1.170	1.188	1.14	1.1198

In accounting for ammonia, Dr. Schniewind states that the total gas liquor produced was found, by difference, to be about 8.17 per cent. of the dry coal, and that the ammonia yield, calculated as sulphate amounted in the test to 1.373 per cent. The nitrogen in the coal amounted to 1.51 per cent., and it was found that 18.5 per cent. of the total nitrogen in the coal was converted into ammonia, but the actual yield at Everett amounted to only slightly more than 1 per cent. in sulphate.

The gas during the test was subject to frequent analyses, and Table LXXVIII gives Dr. Schniewind's computations based on these analyses, the table showing the cumulative results for the average of the gas up to and including each hour of the coking period, the test results, as well as those given above, having been secured in an Otto-Hoffman regenerative oven 33 ft. 6 in. long, 5 ft. 10 in. high, with a taper from 19.5 in. to 22 in., at the Pittsburgh Gas and Coke Company's plant at Glassport, Pa. The figures above the division line between the 14th and 15th hours, are for the surplus gas evolved during the first period of 14 hours and 46 minutes, while the figures below this line represent the oven-fuel gas evolved during the second period of 19 hours and 10 minutes.

Dr. Schniewind calls attention to the fact that the coking process may be divided into three distinct periods, during the first of which the coal is heated up, and a gas very rich in marsh gas and hydrocarbons is evolved; during the second period the coking progresses gradually to the center of the charge, and a coal gas of almost constant quality is produced, while at the beginning of the third period the heat has penetrated from both side walls of the retort to the center of the charge, resulting in a rapid rise in temperature, which results in producing a gas very much richer in hydrogen.[1]

The Otto Coke Oven.—The Otto-Hoffman oven was looked upon as an improvement on the original Coppée oven, and it was the first one in which the Siemens regenerative principle was adopted. In 1894, C. Hilgenstock devised a bottom-fired oven known as the Otto-Hilgenstock oven, its principal feature being the adoption of Bunsen burners throughout the entire length of the oven wall in an attempt to secure a more uniform distribution of heat; the Siemens regenerator was abandoned in this construction, and the sole flue of the oven was used as a recuperator. Figs.

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[1] F. Schniewind, Gas Section, Engineering Congress at Glasgow, 1901.

TABLE LXXVIII.—ANALYSES OF DOMINION COAL GAS PRODUCED IN TEST AT GLASSPORT, PA.

Hours	Analyses of gas produced							Gas free of N_2 and O_2					Sp. gr. of gas		Cal. value of gas		C.P. of gas not freed of CO_2	Total produced per long ton of dry coal		
	C_mH_n	CH_4	H_2	CO	CO_2	O_2	N_2	C_mH_n	CH_4	H_2	CO	CO_2	With	Without	With	Without		Gas, cu. ft.	Cal. value, B.t.u.	Candle feet, gas not freed of CO_2
													N_2 and O_2		N_2 and O_2					
1	5.8	40.3	34.3	6.8	3.9	0.2	8.7	6.4	44.2	37.6	7.5	4.3	0.552	0.510	707	776	18.4	413	292,000	7,599
2	5.8	40.7	34.0	6.8	3.8	0.2	8.7	6.4	44.6	37.3	7.5	4.2	0.550	0.508	709	778	18.4	746	528,000	13,726
3	5.8	41.0	34.0	6.7	3.8	0.2	8.5	6.4	44.9	37.2	7.3	4.2	0.547	0.506	716	773	18.4	1,041	745,000	19,154
4	5.8	41.1	34.1	6.6	3.9	0.3	8.2	6.3	44.9	37.3	7.2	4.3	0.549	0.509	712	778	17.9	1,353	963,000	24,208
5	5.8	41.2	34.2	6.4	3.9	0.3	8.2	6.3	45.0	37.4	7.0	4.3	0.547	0.507	705	770	17.4	1,700	1,209,000	29,552
6	5.7	41.1	34.5	6.3	3.9	0.3	8.2	6.2	44.9	37.7	6.9	4.3	0.543	0.503	709	775	16.8	2,057	1,460,000	34,693
7	5.6	40.9	35.0	6.2	3.9	0.3	8.1	6.1	44.7	38.2	6.8	4.2	0.540	0.500	707	772	16.4	2,399	1,697,000	39,413
8	5.6	40.6	35.4	6.2	3.8	0.3	8.1	6.1	44.3	38.6	6.8	4.2	0.535	0.495	704	769	16.2	2,749	1,936,000	44,488
9	5.5	40.2	36.1	6.2	3.7	0.3	8.0	6.0	43.8	39.4	6.8	4.0	0.529	0.489	701	764	15.8	3,135	2,195,000	49,776
10	5.5	39.8	36.6	6.1	3.7	0.3	8.0	6.0	43.4	39.9	6.7	4.0	0.525	0.483	697	760	15.6	3,479	2,424,000	54,351
11	5.4	39.4	37.2	6.1	3.7	0.3	7.9	5.9	42.9	40.6	6.6	4.0	0.521	0.480	690	752	15.4	3,803	2,640,000	58,628
12	5.4	39.3	37.5	6.1	3.7	0.3	7.7	5.9	42.7	40.8	6.6	4.0	0.518	0.478	692	752	15.2	4,165	2,881,000	63,406
13	5.3	39.1	37.9	6.1	3.6	0.3	7.7	5.8	42.5	41.2	6.6	3.9	0.516	0.476	690	750	15.0	4,528	3,125,000	68,125
14	5.3	38.8	38.2	6.1	3.6	0.3	7.7	5.8	42.2	41.5	6.6	3.9	0.513	0.473	688	748	14.8	4,865	3,349,000	72,270
15	5.2	38.6	38.5	6.1	3.6	0.3	7.7	5.7	41.8	42.0	6.6	3.9	0.511	0.472	685	743	14.6	5,227	3,580,000	76,650
16	5.1	38.4	38.7	6.2	3.5	0.3	7.8	5.6	41.8	42.1	6.7	3.8	0.509	0.468	680	740	14.4	5,591	3,807,000	80,981
17	5.1	38.1	39.0	6.2	3.5	0.3	7.8	5.6	41.5	42.4	6.7	3.8	0.507	0.466	679	739	14.3	5,909	4,011,000	84,765
18	4.8	37.9	39.4	6.2	3.5	0.3	7.9	5.2	41.3	43.0	6.7	3.8	0.506	0.464	676	736	14.2	6,269	4,237,000	89,013
19	4.9	37.8	39.3	6.3	3.5	0.3	7.9	5.3	41.2	42.9	6.8	3.8	0.504	0.462	673	733	14.0	6,614	4,452,000	92,842
20	4.8	37.7	39.6	6.2	3.5	0.3	7.9	5.2	41.1	43.1	6.8	3.8	0.503	0.461	671	731	13.9	6,944	4,658,000	96,340
21	4.7	37.6	39.9	6.2	3.4	0.3	7.9	5.1	41.0	43.4	6.8	3.7	0.501	0.459	669	730	13.7	7,290	4,874,000	100,146
22	4.7	37.4	40.1	6.2	3.4	0.3	7.9	5.1	40.8	43.6	6.8	3.7	0.498	0.456	667	727	13.6	7,634	5,089,000	103,861
23	4.6	37.2	40.4	6.2	3.3	0.4	7.9	5.0	40.6	44.0	6.8	3.6	0.495	0.452	664	724	13.4	8,030	5,330,000	107,940
24	4.5	37.0	40.7	6.2	3.3	0.4	7.9	4.9	40.4	44.3	6.8	3.6	0.492	0.449	660	720	13.3	8,408	5,551,000	111,833
25	4.4	36.8	41.2	6.2	3.2	0.3	7.9	4.8	40.4	44.8	6.8	3.5	0.490	0.448	656	715	13.1	8,728	5,729,000	114,713
26	4.3	36.5	41.5	6.2	3.2	0.3	8.0	4.7	39.8	45.2	6.8	3.5	0.486	0.442	652	711	12.9	9,022	5,886,000	117,006
27	4.2	36.0	42.0	6.2	3.1	0.3	8.2	4.6	39.3	45.9	6.8	3.4	0.482	0.436	648	708	12.8	9,308	6,032,000	118,808
28	4.1	35.6	42.7	6.1	3.1	0.3	8.1	4.5	38.8	46.6	6.7	3.4	0.478	0.433	643	702	12.5	9,583	6,161,000	120,045
29	4.0	35.1	43.2	6.2	3.0	0.4	8.1	4.4	38.3	47.2	6.8	3.3	0.475	0.429	639	698	12.3	9,811	6,265,000	120,911
30	4.0	34.8	43.6	6.2	3.0	0.3	8.1	4.4	38.0	47.6	6.8	3.3	0.472	0.427	635	693	12.2	9,979	6,339,000	121,533
31	3.9	34.5	43.9	6.2	3.0	0.3	8.2	4.3	37.7	47.9	6.8	3.3	0.470	0.423	632	691	11.9	10,123	6,395,000	122,080
32	3.9	34.2	44.2	6.2	2.9	0.3	8.3	4.3	37.4	48.3	6.8	3.2	0.468	0.420	628	687	11.9	10,250	6,441,000	122,537
33	3.9	33.9	44.4	6.2	2.9	0.3	8.4	4.3	37.1	48.6	6.8	3.2	0.467	0.418	626	685	11.8	10,344	6,475,000	122,866
34	3.8	33.9	44.5	6.2	2.9	0.3	8.4	4.2	37.1	48.7	6.8	3.2	0.466	0.417	626	685	11.8	10,390	6,501,000	122,981

89 and 90 show an Otto by-product oven, arranged with one burner for each two vertical flues. The Otto regenerative oven of 1905 combines the Hilgenstock principle in conjunction with a modified regenerator system, longitudinal regenerators being provided under both the charging and discharging wharfs; in this oven reversal periodically takes place in two alternate sections, each equal to one-quarter of the entire length of the side wall; double regenerators are provided, front and rear, one being composed of red and the other of firebrick. The United-Otto oven, as built in the United States, retains certain features of both the old Otto-Hoffman and the new Hilgenstock ovens; the foundations are readily accessible, and may be inspected at any time, while the oven proper retains the half-oven reversal, the regenerators being located under the ovens instead of outside. In the oven as described in Figs. 89 and 90, the gas is introduced into the combustion chamber from a number of burners corresponding to half the number of vertical flues; if the oven should be provided with 32 flues, it would be equipped with 16 burner pipes, these burners entering the combustion chamber from below the ovens. If the burners are numbered from 1 to 16, the method of heating would require that during ½ hour the gas shall enter the combustion chamber from burners 1 to 4 and 9 to 12, the products of combustion traveling through [illegible]cal flues corresponding to burners 5 to 8 and 13 to 16; [illegible] next half hour this process is reversed. The travel [illegible] heat gases in this method of burning seems to place

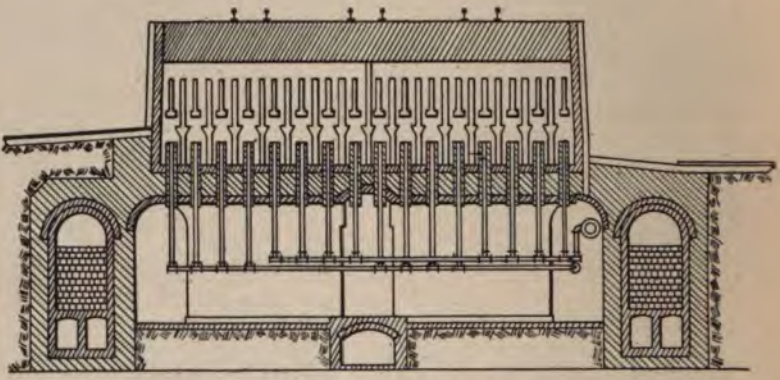

Fig. 89.—Otto by-product coke oven.

them under better control, because the off-heat is withdrawn from each quarter of the oven instead of from each half.

Coppée Retort Coke Oven.—In the COPPÉE oven, Fig. 91, we find two regenerators located longitudinally under the oven battery, so arranged that while the one is preheating the necessary air for combustion during one period, the other is absorbing the heat delivered by the products of combustion. The oven is provided with 30 vertical flues, so arranged as to heat the side walls of the ovens in consecutively alternating sections, each section being equal to one-tenth of the oven length, combustion taking place on a level with the sole of the oven, the products of combustion rising in the first three flues and descending in the next three, and so on. Each side wall of the oven is provided with four gas distributing flues, two at the front and two at the back of the oven, the gas required for combustion being supplied alternately through these flues every half hour. If the flues should be numbered from 1 to 30, and divided into groups of six, the first group would consist of flues 1 to 6; gas for combustion would be delivered from a front flue to flues 1 to 3 during ½ hour, and the products of combustion would pass through flues 4 to 6, this being repeated throughout the entire setting until the next half hour, when the process would be reversed, flues 4 to 6 receiving fuel gas, and flues 1 to 3 carrying away the resultant products. When the products of combustion leave these flues, they are conducted into a collecting flue which runs the entire length of the battery, and then they pass into the regenerators and

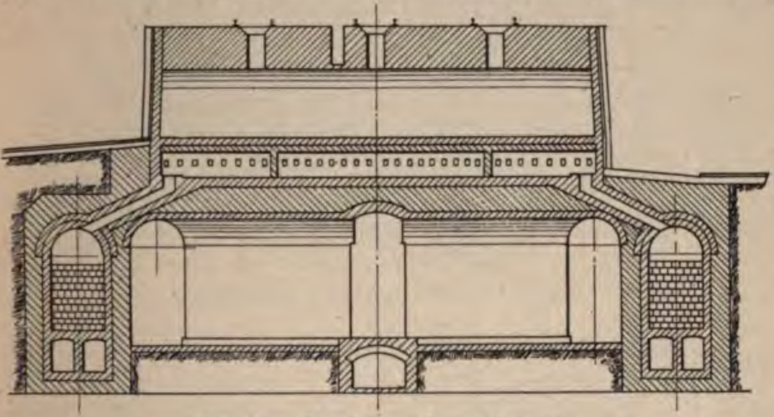

FIG. 90.—Otto by-product coke oven.

from thence to the chimney, a separate chimney being provided for each collecting flue. The regenerated air on its way to the combustion chamber travels in a direction opposite to the travel of the combustion products, reaching the sole flue through oblique ducts which open into the base of the vertical flues.

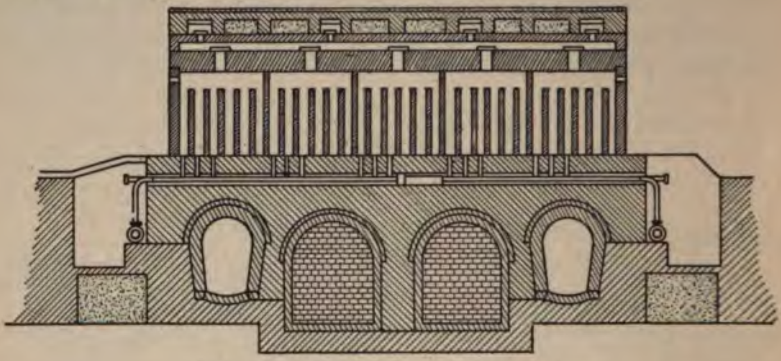

Fig. 91.—Coppée by-product coke oven.

The Muller By-product Oven.—This oven, illustrated in Figs. 92 and 93, is provided with fuel gas by means of pipes located along the sides of the battery, branch pipes, provided with valves, projecting from these mains into the upper as well as into the lower gas ducts on each side of the oven. The oven is so constructed that either regeneration or recuperation may be used. If we assume that the regeneration system is used, the gas will

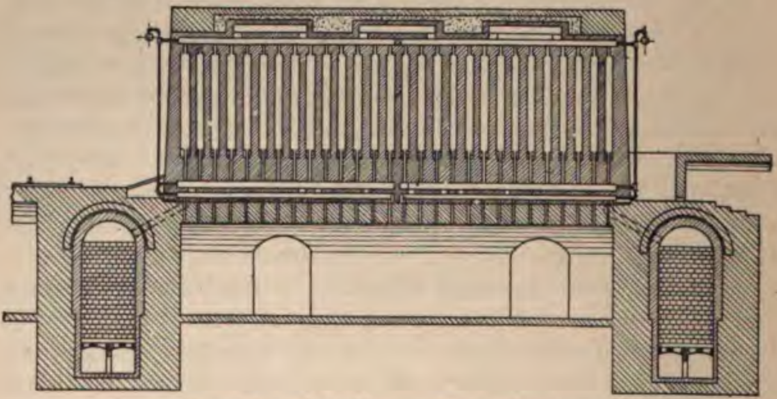

Fig. 92.—Muller by-product coke oven.

pass from the lower channels through the tuyères alternately into half the number of tuyères in one wall, that is, into one side of one or two combined pairs of heating flues. Preheated air also enters from one regenerator through the sole channel and lateral passages through the tuyères, and mixes with the issuing gas, the latter flowing upward when burned. The resulting combustion products are now conducted through the upper passages into the next flues of one or two combined pairs and, since considerable of the heating power is lost on the way upward, each of the flues having downward combustion receives an auxiliary supply of fresh gas and preheated air through the upper tuyères in order to enliven the combustion and to maintain the temperature as high as that in the upward flues, this balance being attained with the expenditure of a small quantity of gas. The waste gases finally descend through the lower tuyères and passages to the sole channels, and pass from there through one or the other of the regenerators to the chimney. The tuyères are so constructed that they convey the gas and air current into the heating flues in such man-

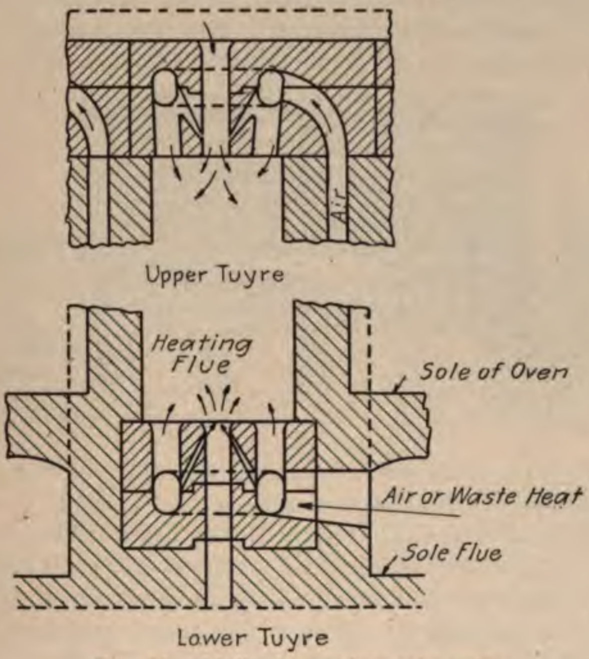

FIG. 93.—Muller by-product coke oven.

ner that their travel will be parallel with, and not baffle each other. In this manner the flame jets are prevented from impinging at any particular place on the wall of the heating flues, and this arrangement also provides an intimate mingling of air and gas, ensuring a rotary movement from the tuyère nozzles into the combustion flues. After a reversal of a suitable three-way cock in the branches of the gas supply pipes and of the reversing valve of the two regenerators alongside the battery, the supply of gas and air is caused to flow in the reverse direction, the flames in the combustion flues also being thereby reversed. If no regenerators are employed for heating the air, the direction of the flow of gas and air and that of the products of combustion in the supply pipes and channels always remains the same.

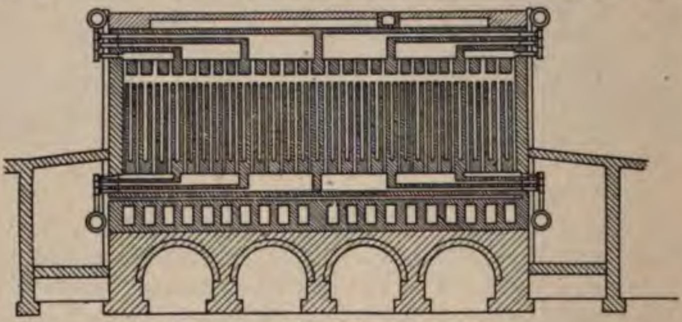

Fig. 94.—Collin retort coke oven.

The Collin Retort Coke Oven.—In the Collin oven, Fig. 94, there is no reversal from one-half of the oven to the other half, because simultaneous heating of the entire side wall of the oven is accomplished by means of 28 vertical main and 27 vertical auxiliary flues, gas for combustion being alternately introduced at the bottom and at the top of the flues. During the first half hour of the operating period, gas and regenerated air come into contact at the bottom of each main flue and, after burning upward, the resultant products descend through the 27 secondary flues and enter into a collecting flue located in the sole of the oven. During the following half-hour period, the gas and air meet at the top of the 28 main flues, the air having been supplied through the 27 auxiliary flues; combustion now occurs downward through the main vertical flues, and the resultant products are again collected in the sole flue. This sole flue leads into a series of rectangular brick passages located beneath the oven battery, and arranged at right angles to the oven chambers, the entire series of chambers being divided into two sections, one of which is supplied with the combustion products for each half hour's heating. From these two sets of passages the products of combustion are led into two regenerators located between the ovens and the chimney. The tier system of end burners is used for introducing the fuel gas, four being provided for each end at the bottom of the oven, and two for each end at the top of the oven, one of the four bottom burners being reversed for the end or outside flue only, the others each supplying the fuel gas required for four or five vertical flues.

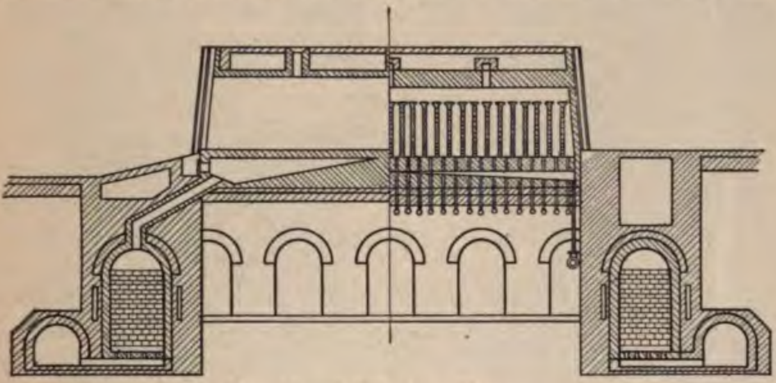

FIG. 95.—Still retort coke oven.

The Still Retort Coke Oven.—The STILL oven is shown in Fig. 95; it is built on the vertical flue principle, and consists of two types, the off-heat type, which does not include by-product recovery, and the regenerative type with by-product recovery. The fuel gas passes from the supply main into a duct located beneath the oven, this duct being provided with a taper from each end to the center of the oven. From this duct the gas is conducted through double nozzles into the vertical flues. The diameters of the nozzle openings vary, that of the upper nozzle being about one-half that of the lower, a vertical space of about 4 in. being provided between the nozzles. The air required for combustion is supplied through a duct which has a taper similar to that of the gas duct, located beneath the oven, and is conducted from here through ports into the vertical flue, being delivered at two points, one at about the level of the gas nozzle and the other at a point located about 15 in. above the other; this is done in order to prevent complete combustion low down in the flue, in this manner avoiding excessive heat at the bottom of the oven. The regenerators are common to all ovens in any one battery, and they extend along the entire length of the battery on each side.

The Von Bauer Coke Oven.—The chief claim made for this oven, Figs. 96 and 97, is a uniformity of heating conditions, the builders further claiming that reversals in the direction of the currents of the products of combustion and of the preheated air cause violent changes in temperatures, these changes resulting detrimentally to both the structure and the products obtained. This oven therefore does not embody any reversal features, differing thereby from nearly all other constructions. Independent and direct gas supplies are provided in the upper part of the oven for carrying the gas to different parts of the flue, these supplies being controlled by valves located immediately opposite the upper parts of the partition walls. Air for combustion is supplied from the spaces formed by the foundation arches under the ovens, these arch spaces serving for the preliminary heating of the air supply as well as for the cooling of the oven foundations. The air

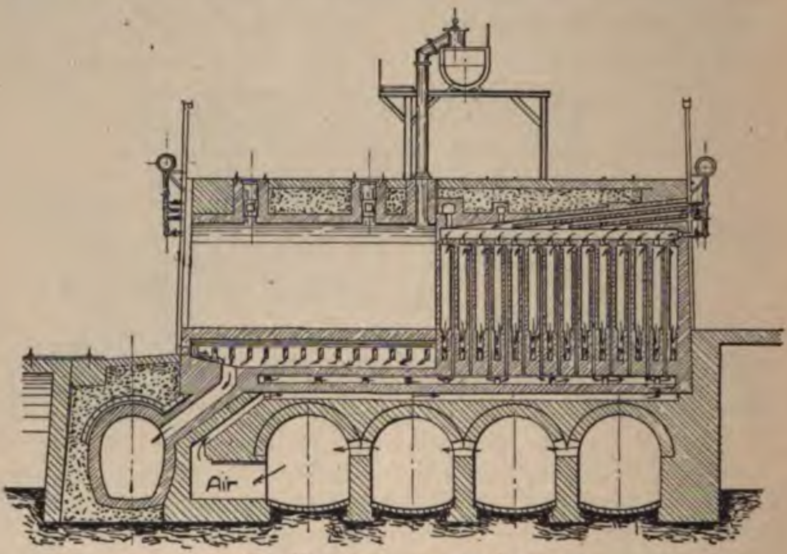

FIG. 96.—Von Bauer retort coke oven.

from these chambers enters the distributing duct below the oven, where it is heated to about 1100°F. (595°C.), and rises from here through uptakes located in the walls of the coking and combustion chambers, it being claimed that the air temperature is thereby increased to 1920°F. (1049°C.). The gases of combustion have a downward course and pass from the heating flue into a duct leading to the chimney.

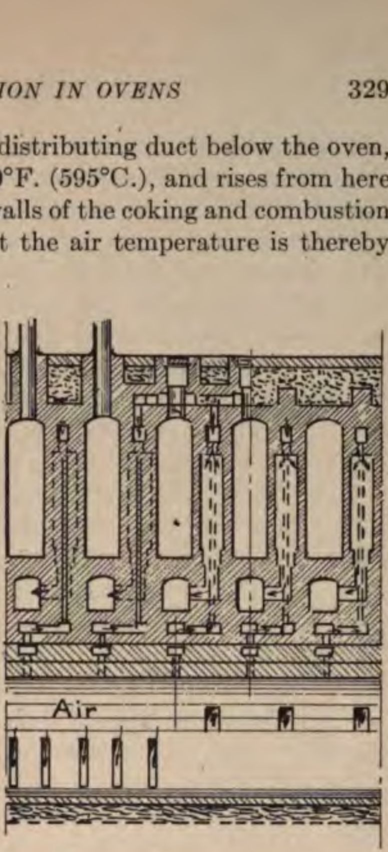

Fig. 97.—Von Bauer retort coke oven.

The Koppers By-product Coke Ovens.—This oven, Fig. 98, is one of that type in which the heating is performed on each half of the oven alternately, the reversal taking place every half hour. Fuel gas is taken from the gas main and admitted into a firebrick gas duct located below the vertical flues, this duct being provided with an individual opening for each vertical flue, the gas entering the flue through a fire-clay nozzle located in the gas duct opening, the size of the opening in the nozzle depending on its location in the oven. The air required for combustion is led from the regenerators into each vertical flue through individual openings in the roof of the regenerator chamber; this air meets the fuel gas

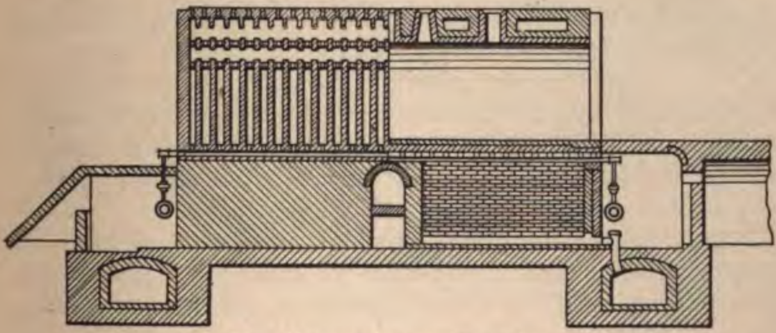

Fig. 98.—Koppers by-product coke oven.

at a point about 10 in. above the gas nozzle in the vertical flue, the products of combustion entering a horizontal flue from whence they are drawn down through the vertical flues on the other half of the oven, thence entering the regenerators and then passing on to the chimney. In order to overcome the tendency of the gases to pass by means of the shortest route to the chimney, which would lead to the bulk of the hot gases being taken away in the center flues, thereby producing an ununiform heating condition, individual draft regulation is provided through the medium of a sliding brick damper located in each vertical flue, this damper controlling the size of the opening leading into the horizontal flue. The top of the oven is provided with an opening over each vertical flue for the purpose of providing access to this brick damper, the opening being normally closed by means of a firebrick plug. The air supply for each individual oven is also regulated by means of a sliding valve, and the sliding brick damper above referred to provides individual regulation of the supply of air at each vertical flue during the combustion period.

A test[1] made on Koppers ovens at the plant of the Maryland Steel Company, Sparrows Point, Md., gave the following results:

The coal carbonized was of two kinds, (1) West Virginia coal from Underwood, W. Va., (2) Pennsylvania coal from the Pittsburgh district, the average analyses of which gave:

	West Virginia	Pittsburgh
Volatile matter	37.34	35.19
Fixed carbon	55.79	56.76
Ash	6.69	8.05
Sulphur	1.44	1.24
Phosphorus	0.013	0.014

The tar produced amounted to 14.3 gal. per net ton of coal, and the ammonia to 6.94 lb., equivalent to 27.76 lb. of sulphate. The gas results are given in Table LXXIX.

The Koppers Company used the Barnhart modification of the Hempel apparatus for gas analyses, all nitrogen determinations being checked, while the Gas Company used the Elliott apparatus. The calorific values were determined by the use of a Junkers calorimeter.

The Koppers Company also builds a small oven unit to meet the demand for a system suitable for small gas works, and in which a large yield of gas and by-products may be obtained with

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[1] C. J. Ramsburg, American Gas Institute, 1914.

TABLE LXXIX.—GAS RESULTS

Date, Dec., 1913	No. of ovens charged	Average coking time	Coal charged, pounds	Coal charged per oven	Cor. gas made in cubic feet	Cor. yield, per pound	Cal. value, B.t.u.	Cal. multiple	C. P. New Sugg. *D*	Candle feet	Coal through ¼-in. mesh, per cent.	Wt. of cu. ft. of dry coal
	West Virginia Coal. Dec. 2 to 6, 1913, inclusive											
2	7	20¼	215,000	30,730	1,090,551	5.07	616	3,113	17.15	....	43.50	50.14
3	6	20¼	187,200	31,200	1,119,450	5.98	631	3,773	17.60	....	41.00	50.91
4	7	20⅕	212,300	30,300	1,159,160	5.46	621	3,391	15.82	....	41.30	49.48
5	7	20⅕	221,600	31,667	1,132,380	5.11	619	3,163	16.02	....	43.40	51.65
6	7	20	213,200	30,460	1,151,280	5.40	634	3,424	15.49	....	48.40	49.66
Average	6.8	20⅕	209,880	30,860	1,130,560	5.39	624	3,363	16.42	88.50	43.50	50.37
	Pittsburgh Gas Coal. Dec. 8 to 20, 1913, inclusive											
11	7	20	220,400	31,490	1,179,140	5.35	614	....	15.52	....	33.50	51.36
12	7	20	219,700	31,380	1,184,180	5.39	608	....	15.34	....	47.10	51.19
13	7	20	223,300	31,900	1,165,620	5.22	612	....	16.90	....	42.20	52.05
14	7	20	219,600	31,370	1,210,000	5.51	612	....	17.29	....	47.90	51.18
15	7	20	228,400	32,630	1,178,540	5.16	600	....	16.01	....	46.50	53.20
16	7	20	222,100	31,730	1,150,480	5.18	598	....	15.23	....	49.30	51.75
17	7	20	226,000	31,280	1,213,600	5.37	607	....	15.19	....	53 50	52.50
18	7	20	223,300	31,900	1,221,450	5.47	596	....	15.21	....	43.20	52.00
19	7	20	218,400	31,200	1,179,300	5.40	608	....	15.17	....	49.70	50.83
20	7	20	210,400	30,060	1,153,000	5.48	593	....	15.39	....	38.80	49.33
Average	7	20	221,160	31,580	1,183,530	5.35	605	3.237	15.72	84.10	43.40	51.34

TABLE LXXX.—GAS ANALYSES

Made by Koppers Company

West Virginia Coal

Dec., 1913	CO_2	O_2	Ill.	CO	CH_4	H_2	N_2	B.t.u. per cu. ft.
2	1.85	1.25	4.35	7.10	32.30	47.50	5.65	616
3	1.70	1.35	4.65	7.10	33.20	46.60	5.40	631
4	1.40	0.60	4.65	7.05	31.65	50.45	4.20	621
5	1.70	0.80	4.50	7.15	30.75	49.40	5.70	619
6	1.50	0.65	4.10	6.75	32.25	48.25	6.50	634
Made by Con. G. E. L. and P. Company after pumping to Baltimore								
2	1.7	0.4	4.7	7.0	33.7	50.9	1.6	621
3	1.5	0.7	5.4	6.7	35.1	49.3	1.3	650
4	1.3	0.7	4.7	6.6	31.9	53.6	1.2	619
5	1.5	0.7	4.5	6.6	30.8	55.6	0.3	629
6	1.2	0.9	4.3	6.5	32.8	52.4	1.9	631
Pittsburgh Coal Made by Koppers Company								
11	1.25	0.65	3.60	6.80	30.85	50.05	6.70	614
12	1.35	1.05	3.25	6.55	29.70	51.15	6.95	608
13	1.15	0.75	3.65	6.85	30.95	52.05	4.60	612
14	1.30	0.95	4.05	6.55	31.25	52.40	3.50	612
15	1.35	0.90	3.65	6.60	31.45	51.65	4.40	600
16	1.40	1.00	3.35	6.80	30.50	52.35	4.60	598
17	1.00	0.90	3.65	6.75	30.95	51.25	5.50	607
18	1.30	1.05	3.95	6.95	31.70	50.30	4.75	596
19	1.45	0.90	3.60	6.90	30.90	52.40	3.85	608
20	1.40	1.05	3.10	7.35	21.10	51.75	4.75	593
Made by Con. G. E. L. and P. Company after pumping to Baltimore								
11	1.8	0.6	3.2	6.4	31.7	54.4	1.9	631
12	1.4	0.4	4.2	6.3	33.0	51.6	3.1	616
13	1.4	0.2	3.6	6.2	30.9	55.8	1.9	598
14	1.2	0.8	3.7	6.2	31.4	54.5	2.2	615
15	1.2	0.4	3.8	6.0	32.3	54.3	2.0	601
16	1.6	0.5	3.6	6.0	31.4	54.6	2.3	615
17	1.6	0.5	3.7	6.5	32.0	53.8	1.9	608
18	1.6	0.5	4.0	6.1	30.8	56.3	0.7	607
19	1.3	0.7	3.4	6.4	32.8	53.0	2.4	593
20	1.3	0.4	3.3	6.5	31.3	52.4	4.8	586

a minimum of labor and investment costs. The oven battery, consisting of from 10 to 16 ovens of 6 tons capacity, is so designed that it is claimed no labor is required at night other than a watchman and the engineer who attends the machinery in the by-product house. Charging and discharging is accomplished during the day, and the ovens may be shut down over Sundays and holidays, it being claimed that this will not result in damage to the oven, nine men being required during 24 hours' operation for taking care of the entire plant. It is also claimed that with a first-class coal a 60-ton 10-oven plant will produce 30 tons of good, hard coke, 3 tons of breeze, 600 gal. of first quality tar, ¾ ton of ammonium sulphate and 600,000 to 650,000 cu. ft. of 600 B.t.u. gas of about 16 candles measured on a Sugg. "D" burner. The quality of the coke produced will, of course, depend upon the character of the coal used, but if the plant operates to produce a 600 B.t.u. gas, the resultant coke should be of first quality for domestic use as well as for many metallurgical purposes. It is claimed that the tar will be of low viscosity, containing not more than 6 per cent. of free carbon and a minimum of unsulphoniable ingredients, while a large percentage of creosote oil with a minimum of pitch should be present. The first installation of this type of oven was erected at Innsbruck, Austria, in 1908, and the following results are taken from the report of the director for 1913.

Coking time, hours,	24.0
Coal charged per year, net tons	12,665.5
Volatile matter in coal, per cent.	32.0
Gross coke produced, net tons	9,120.98
Tar produced, gallons	169,490.0
Ammonium sulphate produced, net tons	128.75
Production per net ton of coal gasified:	
Gas at 30 in. 60°F., cubic feet	11,024.0
Coke, pounds	1,440.0
Tar, gallons	13.38
Ammonium sulphate, pounds	20.3

Coke balance:	
Coke produced per net ton of coal, pounds	1,440
Coke used for firing, pounds	254
	1,186
Gas results at 30 in., 60°F. saturated:	
Per pound of coal, cubic feet	5.512
B.t.u. per cubic foot	595.0
B.t.u. per pound of coal	3,279.6

Temperature Conditions in Coke Ovens.[1]—PROF. OSKAR SIMMERSBACH, of Breslau, Germany, took up the work published by HILGENSTOCK in 1902, as regards temperature conditions in coke ovens, and his present statements refer to the temperature conditions in the center of the oven, at the machine end, and at the quenching end. SIMMERSBACH's experiments were made on a KOPPERS oven, heated with coke-oven gas, the oven having the following dimensions:

Length of oven	10,000 mm.	32 ft. 9¾ in.
Mean width	500 mm.	19⅞ in.
Useful height	2,300 mm.	7 ft. 6 in.
Crown	100 mm.	4 in.

The oven was charged with 8.8 tons (German) of coal, the coal analyzing 22.70 per cent. volatile matter, 6.32 per cent. ash, and 12.1 per cent. water, so that the charge consisted of 7735 kg. (17,017 lb.) of coal and 1065 kg. (2343 lb.) of water. The coke yield amounted to 79.86 per cent., or 6177 kg. (13,590 lb.). In order to insert the pyrometers, the lids of the three trunnel heads were perforated and two pyrometers were inserted in each. The one set of pyrometers, shown at 1, 3 and 5 in Fig. 99, had a length of 2.35 meters (92½ in.) and entered the coal so that the junction lay 1 meter (39⅜ in.) above the sole of the oven, while the second set of pyrometers, shown at 2, 4, and 6, had a length of 85 cm. (33½ in.) and ended 10 cm. (4 in.) below the top of the oven. All six pyrometers were connected to a SIEMENS & HALSKE registering galvanometer provided with a switch for six measuring points. Contact was made with a different measuring point every 2 minutes, thus each element was caused to register the temperature once in every 12 minutes. The various temperatures were read and corrected, the corrected figures being given in Table LXXXI.

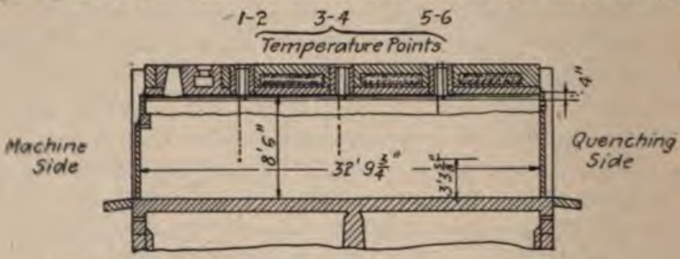

FIG. 99.—Location of pyrometers, Simmersbach tests.

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[1] Stahl und Eisen, June 14, 1914, page 954.

TABLE LXXXI.—TEMPERATURE IN COKE OVEN RECORDED BY SIMMERSBACH

1	2		3		4		5		6		7	
	In the coal						Above the coal					
Time in hours	Machine side		Middle of oven		Quenching side		Machine side		Middle of oven		Quenching side	
	C.°	E.°	C.°	F.°	C.°	F.°	C.°	F.°	C.°	F.°	C.°	F.°
0.0	10	50	10	50	10	50	730	1346	750	1382	760	1400
0.5	10	50	10	50	10	50	730	1346	750	1382	730	1346
1.0	10	50	10	50	10	50	720	1328	750	1382	720	1328
1.5	10	50	10	50	10	50	730	1346	760	1400	710	1310
2.0	10	50	10	50	10	50	730	1346	750	1382	700	1292
2.5	100	212	10	50	10	50	730	1346	745	1373	700	1292
3.0	100	212	100	212	10	50	730	1346	740	1364	690	1274
3.5	100	212	100	212	100	212	720	1328	740	1364	690	1274
4.0	100	212	100	212	100	212	720	1328	740	1364	690	1274
4.5	100	212	100	212	100	212	730	1346	750	1382	690	1274
5.0	100	212	100	212	100	212	735	1355	750	1382	690	1274
5.5	100	212	100	212	100	212	735	1355	755	1391	690	1274
6.0	100	212	100	212	100	212	735	1355	755	1391	690	1274
6.5	100	212	100	212	100	212	740	1364	755	1391	695	1283
7.0	100	212	100	212	100	212	740	1364	755	1391	695	1283
7.5	100	212	100	212	100	212	740	1364	755	1391	700	1292
8.0	100	212	100	212	100	212	745	1373	760	1400	700	1292
8.5	100	212	100	212	100	212	750	1382	765	1409	705	1301
9.0	100	212	100	212	100	212	750	1382	770	1418	705	1301
9.5	100	212	100	212	100	212	755	1391	775	1427	710	1310
10.0	100	212	100	212	100	212	755	1391	775	1427	710	1310
10.5	100	212	100	212	100	212	755	1391	775	1427	710	1310
11.0	100	212	100	212	100	212	760	1400	775	1427	715	1319
11.5	100	212	100	212	100	212	765	1409	780	1436	715	1319
12.0	100	212	100	212	100	212	775	1427	785	1445	715	1319
12.5	120	248	100	212	100	212	775	1427	785	1445	715	1319
13.0	130	266	100	212	100	212	775	1427	790	1454	720	1328
13.5	135	275	115	239	100	212	775	1427	790	1454	720	1328
14.0	150	302	130	266	100	212	775	1427	795	1463	730	1346
14.5	200	392	180	356	100	212	775	1427	795	1463	730	1346
15.0	210	410	190	374	100	212	775	1427	795	1463	730	1346
15.5	215	419	190	374	120	248	775	1427	800	1472	730	1346
16.0	260	500	200	392	150	302	790	1454	805	1481	740	1364
16.5	330	626	210	410	240	464	790	1454	805	1481	740	1364
17.0	370	698	250	482	270	518	795	1463	810	1490	745	1373
17.5	420	788	310	590	310	590	800	1472	810	1490	745	1373
18.0	440	824	330	626	330	626	810	1490	820	1508	750	1382
18.5	480	896	370	698	380	716	810	1490	820	1508	750	1382
19.0	520	968	430	806	410	770	815	1499	825	1517	750	1382
19.5	590	1094	440	824	490	914	815	1499	825	1517	755	1391
20.0	630	1166	410	770	590	1094	820	1508	830	1526	760	1400
20.5	760	1400	445	833	640	1184	820	1508	830	1526	760	1400
21.0	810	1490	570	1058	570	1058	820	1508	825	1517	760	1400
21.5	865	1589	680	1256	545	1013	825	1517	830	1526	765	1409
22.0	900	1652	725	1337	560	1040	825	1517	830	1526	765	1409
22.5	930	1706	770	1418	565	1049	830	1526	835	1535	770	1418
23.0	955	1751	800	1472	590	1094	835	1535	840	1544	775	1427
23.5	980	1796	910	1670	560	1040	835	1535	840	1544	780	1436
24.0	995	1823	940	1724	565	1049	840	1544	845	1553	780	1436
24.5	1010	1850	890	1634	665	1229	850	1562	845	1553	785	1445
25.0	1030	1886	870	1598	750	1382	855	1571	850	1562	785	1445
25.5	1045	1913	925	1697	780	1436	860	1580	855	1571	790	1454
26.0	1060	1940	960	1760	810	1490	865	1589	860	1580	795	1463
26.5	1065	1949	965	1769	865	1589	865	1589	860	1580	795	1463
27.0	1070	1958	995	1823	870	1598	865	1589	860	1580	800	1472
27.5	1075	1967	1025	1877	875	1607	865	1589	860	1580	805	1481
28.0	1080	1976	1055	1931	880	1616	865	1589	860	1580	810	1490
28.5	1090	1994	1090	1994	900	1652	870	1598	865	1589	810	1490
29.0	1090	1994	1120	2048	920	1688	870	1598	865	1589	810	1490

The coal was coked during a period of 29 hours, and an analysis of the coke showed:

	Per cent.
Ash	8.91
Volatile	2.56
Carbon	88.53
	100.00

and the analyses of the gas, made every hour, is given in Table LXXXII.

Table LXXXI shows that the temperature in the coal increases more rapidly in both the middle of the oven and on the machine side than it does on the quenching side; again, the temperature throughout the entire period of carbonization is lower on the quenching side than at any other portion of the oven, this difference being probably due to the fact that the oven is wider at the quenching side than at any other portion, and, therefore, contains more coal and water burden at this point. An examination of the governing conditions shows that the evaporation of the water is not complete until after the fifteenth hour on the quenching side, while twelve hours suffice for the remainder of the oven.

An examination of the gas analysis, Table LXXXII, shows that the carbon dioxide content decreases from its maximum of 3.3 per cent. at the beginning of the carbonizing period to 1.1 per cent. at the nineteenth hour, remaining practically constant after that.

The benzene content decreases from 1.8 per cent. at the beginning to zero at the end, while the decrease in the ethylene content is from 4 to 0.3 per cent.

The carbon monoxide increases from 0.9 to 5.8 per cent., while the methane content decreases from 36.65 per cent. to 4.7 per cent. between the beginning and the end of the carbonizing period. The hydrogen content increases from 42.5 per cent. to 70 per cent.

The calorific value of the gas suffers a decrease from 612 B.t.u. per cubic foot at the beginning, to 274 B.t.u. at the end, a difference of 55 per cent., the most marked difference, 38 per cent., appearing between the twenty-first hour and the end; this decrease in calorific value is coincident with the decrease in benzene, ethylene, and methane content, and with the cessation of water evaporation.

TABLE LXXXII.—ANALYSES OF GAS RECORDED BY SIMMERSBACH

Hour	2	3	4	5	6	7	8	9	10	11
CO_2, carbon dioxide, volume per cent	3.3	3.0	2.3	2.5	2.2	2.0	1.4	1.9	2.3	1.3
C_6H_6, Benzene, volume per cent	1.8	1.5	1.1	1.1	1.0	0.8	0.6	0.5	0.35	0.35
C_2H_4, Ethylene, volume per cent	4.0	3.5	3.1	3.2	2.8	2.5	2.5	2.0	1.75	2.05
O_2, Oxygen, volume per cent	1.05	0.8	0.6	0.6	0.5	0.5	0.2	0.2	0.2	0.1
CO, Carbon monoxide, volume per cent	0.9	1.9	2.9	2.8	3.0	3.95	3.4	3.1	2.8	2.8
CH_4, Methane, volume per cent	36.65	36.1	34.5	34.5	33.6	32.4	33.65	33.45	31.2	32.4
H_2, Hydrogen, volume per cent	42.5	44.6	48.8	47.8	50.1	50.75	53.65	50.55	47.1	51.5
N_2, Nitrogen, volume per cent	10.0	8.6	6.7	7.5	6.8	7.1	3.6	8.3	14.3	9.5
Lower heat value of gas, cal	5402	5271	5086	5071	4971	4802	4906	4797	4324	4583
(0°C., dry, 760 mm.) B.t.u	612	597	576	574	563	544	556	543	490	519
Hour	12	13	15	17	19	21	23	25	27	Fuel gas
CO_2, carbon dioxide, volume per cent	1.5	2.0	1.9	1.8	1.1	1.05	0.8	0.7	1.0	2.1
C_6H_6, Benzene, volume per cent	0.35	0.3	0.25	0.1	...	...	...	...	...	...
C_2H_4, Ethylene, volume per cent	1.85	1.8	1.8	1.9	1.2	1.0	0.6	0.3	0.3	1.8
O_2, Oxygen, volume per cent	0.2	0.2	0.15	0.25	0.15	0.05	0.05	0.1	0.3	0.6
CO, Carbon monoxide, volume per cent	4.15	4.1	3.9	4.15	4.7	4.0	3.8	4.9	5.8	4.2
CH_4, Methane, volume per cent	33.4	32.45	33.2	30.6	26.1	21.15	18.95	12.2	4.7	26.2
H_2, Hydrogen, volume per cent	50.65	49.75	53.4	51.6	55.75	58.95	61.9	67.0	70.0	51.7
N_2, Nitrogen, volume per cent	7.9	9.4	5.4	8.6	11.0	13.8	13.9	14.8	17.9	15.4
Lower heat value of gas, cal	4660	4530	4665	4369	3978	4487	3412	2960	2419	3964
(0°C. dry, 760 mm.) B.t.u	528	513	528	495	450	508	386	335	274	449

In the matter of the decomposition of coke-oven gas, MR. E. BURY[1] states that he placed a 10-in. pipe vertically in the charge of a coke oven, the lower end of the pipe penetrating some inches into the coal; throughout the carbonizing period the gases passed steadily through this central pipe, the quantity being greatest after carbonization had proceeded about 4 in. toward the center of the charge, or, when the gases came within the scope of the pipe. Analysis of the gases from the pipe evolved 4 hours after carbonization had begun showed them to contain 13.8 per cent. of illuminants and 65 per cent. of methane, the average amount of marsh gas in the bulk coke-oven gas being from 33 to 34 per cent. In view of this enormous hydrocarbon content in the gases as generated in the coal and removed by the pipe before they have time to decompose, there can, in MR. BURY's opinion, be little doubt that, in practice, the gases which pass through the center of the charge are subsequently decomposed on the hot crown of the oven.

PROF. SIMMERSBACH[2] states that the decrease in the heating value of coke-oven gas when preheated in the gas chambers of a SIEMENS-MARTIN furnace is from 25 to 33 per cent., instead of about 4 per cent., as has been supposed. As an example, he calls attention to a laboratory experiment in which the same gas was passed over heated refractory material for 11 to 14 seconds at successively higher temperatures, the deposited carbon being burnt off after each experiment, with the following results:

	Before heating	After heating to				
		800°C. 1472°F.	900°C. 1652°F.	1000°C. 1832°F.	1100°C. 2012°F.	1220°C. 2228°F.
Carbon dioxide..........	3.4	2.1	1.3	1.0	0.7	0.2
Oxygen.................	0.9	0.5	0.3	0.2	0.2	0.1
Carbon monoxide.......	6.3	11.0	12.0	12.9	13.5	14.4
Hydrocarbons...........	2.7	2.0	0.2	0.1	0.0	0.0
Methane...............	33.2	27.6	20.2	17.3	15.6	5.5
Hydrogen..............	42.6	48.6	58.8	63.7	66.1	75.7

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[1] Gas World, July 3, 1909, page 4.

[2] Jnl. Soc. of Chem. Ind., Feb. 28, 1913, page 186.

The temperature to which the gas is heated in the regenerator of a Siemens-Martin furnace is from 1000°C. to 1060°C. (1832°F. to 1940°F.). The decrease in heating value per unit volume is accompanied by an increase in the volume of gas, and therefore by an increase in the speed of the gas current, and this in turn leads to the extension of the zone of combustion beyond the furnace chamber. Simmersbach further states that the decomposition of the gas by heat involves the following reactions:

$$\begin{aligned} CH_4 + 3CO_2 &= 4CO + 2H_2O. \\ CO + H_2O &= CO_2 + H_2. \\ CH_4 + H_2O &= CO + 3H_2. \end{aligned}$$

Also, a gas containing 68.3 per cent. of methane before heating contained 58.5 per cent. at 800°C. (1472°F.), 54.3 per cent. at 900°C. (1652°F.), 32.6 per cent. at 1000°C. (1832°F.), 7.7 per cent. at 1100°C. (2012°F.), and 5.8 per cent. at 1200°C. (2192°F.), and that the decomposition of ethylene begins at 500°C. (932°F.) and is practically complete between 900° and 1000°C. (1652° and 1832°F.).

Further experiments by Simmersbach[1] showed that the heating of coke-oven gas reduces the calorific value according to the temperature employed, as will be seen from the following:

	Before heating, per cent.	Heated to 900°C. (1652°F.), per cent
CO_2	3.4	2.8
CO	5.6	6.0
H_2	47.7	43.5
CH_4	30.3	25.3
Calorific value	513 B.t.u.	444 B.t.u.

	Before heating, per cent.	Heated to 1065°C. (1949°F.), per cent.
CO_2	2.4	3.2
C_mH_n	2.0	0.2
CO	4.2	9.6
H_2	51.1	35.7
CH_4	30.2	20.0
Calorific value	515 B.t.u.	382 B.t.u.

The composition of three samples of gas before and after heating to different temperatures is shown by the following:

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[1] Jnl. für Gasbeleuchtung, Dec. 13, 1913, page 1242.

	Before heating			After heating to		
				750°-760°C. 1382°-1400°F.	900°-940°C. 1652°-1724°F.	1000°-1060°C. 1832°-1940°F.
Sample No.	1	4	6	1	4	6
CO_2	5.0	4.0	5.6	4.0	1.6	0.4
CmHn	1.6	1.4	1.4	0.8	....	....
O_2	0.6	1.2	0.6	0.6	....	....
CO	10.8	11.4	10.0	12.6	20.0	19.2
H_2	49.2	54.8	45.0	50.2	58.5	60.0
CH_4	21.3	16.4	19.8	20.5	7.8	4.8
N_2	11.5	10.8	17.6	11.3	12.1	15.6

Further investigations, made by maintaining the coke-oven gas at increasing temperatures, showed the following effect:

Before heating	Tem. C.° Tem. F.°	810 1490	900 1652	1010 1850	1100 2012	1210 2210
	Duration sec.	22	19	14	14	16
3.4	CO_2	2.6	2.5	1.1	0.6	0.0
0.7	O_2	0.5	0.4	0.3	0.2	0.0
7.9	CO	8.1	8.7	9.0	11.2	11.8
2.6	CmHn	2.2	0.8	0.2	0.0	0.0
31.7	CH_4	28.9	25.1	24.8	16.0	6.8
42.8	H_2	49.3	54.6	57.1	65.1	74.9

The results of exposing a gas rich in methane to a firebrick wall covered with deposited carbon is shown by the following:

	Original composition	After heating to				
		800°C. 1472°F.	900°C. 1652°F.	1000°C. 1832°F.	1100°C. 2012°F.	1200°C. 2192°F.
O_2	0.3	0.2	0.1	0.0	0.0	0.0
CO	0.3	0.7	0.8	0.8	0.5	0.5
CmHn	0.7	0.4	0.3	0.1	0.0	0.0
CH_4	68.3	58.5	54.3	32.6	7.7	5.8
H_2	30.4	40.2	44.3	65.4	91.6	93.5

Conclusions.—As regards the constructive features of coke ovens, Mr. C. A. Meissner states that regenerative ovens are more advantageous for installation at steel works, this latter construction giving a larger quantity of surplus gas than would a recuperator construction, this surplus gas being a very valuable asset to the steel plant, as this gas has a calorific value of from 400 to 650 B.t.u. depending on the character of coal used. It can be used for many heating purposes, and its value at one of the plants of the U. S. Steel Corporation is equivalent to approximately 500,000 net tons of coal per year, based on 40,500 cu. ft. of surplus gas being equivalent to 1 net ton of coal. The gas is used for illuminating and can be piped long distances for this purpose, while in Europe it is used quite extensively in gas engines. Experiments have also been made regarding its use alone in open-hearth furnaces, and in mixing it with blast-furnace gas, the latter method having been largely adopted in certain plants of Europe, and, according to MEISSNER, this system will ultimately receive consideration in this country. As regards the location of the regenerators, whether they shall be placed under each oven, or whether they shall extend longitudinally along the entire battery of ovens, is a matter of discussion. MEISSNER states that he has seen many ovens of the latter type which gave uniform combustion temperatures, but, on the whole, he prefers the type having regenerators under each oven, as, in his opinion, the temperature in such regenerators can be more readily controlled. Also, in a battery of ovens of this type, operations are not liable to be influenced by the action of any oven either not operating properly or shut down for repairs or for other reasons.

Vertical and horizontal flue systems are both in operation in this country; the horizontal flue system ovens are doing very good work and they have the merit of simplicity, and, according to MEISSNER'S observations, the combustion of gas and air in them has been controlled just as completely as in the vertical flue ovens, thus producing as uniform temperatures as can be obtained with vertical flues. MEISSNER further states that the combustion in, as well as the accessibility to, the vertical flue oven seems to be more easily and better controlled in those types whose flues open to the top of the oven than in those whose flues are accessible only from below, or those which are entirely closed in; the great objection to many of these latter types being the possibility of something going wrong with a flue at any time which might entail the stopping of the oven and the tearing out of some of the brick-work, while this would not be necessary in types where the flues open to the top of the retort, or in ovens of the horizontal flue type.

This condition in some ovens of both the newer and the older types leads to a complicated system of combustion and carrying off the waste gases. Claims are at times made for these types to the effect that they will give a more perfect combustion, due to their flue arrangement, but, on the other hand, it is hard to see why the standard vertical flue, and the standard horizontal flue types should not give perfect combustion and temperature regulation of both flues and oven walls. Careful attention should be paid to the construction of the flue walls, and they should be laid up in such manner that leakage, or the short-circuiting of air and products of combustion from one flue to the other cannot take place. This is usually caused at the very beginning, due to expansion, when the oven is being heated up, and a flue system which is complicated, and which has more than the absolutely necessary passages upward or alongside the oven walls, subjects itself to danger from this source.

Meissner further states that the air ports, and the consequent draft, are very important matters, as the coking time in great measure will depend on the area of these ports and flues. A short coking period requires considerable area in order to enable the operator to burn sufficient gas to give him the temperatures he desires. Earlier constructions erred in this particular, with the result that frequently not sufficient draft was provided to give the necessary rapid combustion required for high heats. This can be corrected through increased stack draft, or by the application of induced draft, but in constructing new ovens this condition should be given every attention. Meissner is of the opinion that one draft stack should be supplied to every battery of coke ovens, the flues having sufficient area to take care of atmospheric or other fluctuations.

Experience has shown that a taper of about 4 in. in the width of the oven from the pushing end to the discharging end is of great importance, especially where swelling coal is used, but even where this is not the case, a taper always has the advantage of presenting less resistance to the movement of the coke. After the first effort of putting the mass of coke in motion, the actual pushing becomes easier, less small coke is produced, and the walls of the oven are conserved. Practically all the different coke oven constructors have now arranged their heating flues in such manner that the necessary amount of heat penetrates into the coal mass at the wide end of the oven in the same time as it does in the narrow end.

Modern by-product ovens should be provided with self-sealing doors, in order to prevent leakage and to conserve labor and material. The new plant of the Western United Gas and Electric Company, at Joliet, Ill., is provided with self-sealing doors, the first to be applied satisfactorily in this country, although they have been used in Vienna, Austria, for some years without trouble. These doors are made of flexible steel, heavily bricked, and they close upon a water-cooled pipe welded onto the door frame. Nine to 10 per cent. of nitrogen has been considered fair coke-oven practice, but in Vienna, with the use of these self-sealing doors, the nitrogen content has been brought down to 4 per cent.

W. S. BLAUVELT[1] states that in thermal efficiency and in total yield of gas, the by-product oven compares favorably with the best gas benches, and that the development of that type of oven which may be heated by burning producer gas or oven gas at will, will provide a simple and satisfactory means of adjusting the daily make of gas to the demand without changing the rate of carbonization. By the ordinary method of charging and leveling a retort oven, the level of the charge on top is always at fixed elevation, depending on the location of the leveling ram, and with the horizontal flue oven it is possible to have the top flue at any temperature required; therefore, the possibility exists of varying the location of the top horizontal flue in the design of the oven, so that this flue may have a distinct location relative to the level of the top of the charge of coal, this relation having a distinct bearing on the quality of the coke made at that point. In England and on the Continent the plan is generally followed of leveling the charge of coal at an elevation from 6 to 10 in. above the top of the top flue, so that the carbonization of the coal in the top of the oven is carried on very largely by the heat which ascends from the lower and hotter portions of the oven. This is probably a distinct advantage in the manufacture of foundry coke, because the top portion of the charge of coke is usually rejected in loading this class of coke.

[1] Proc. A. G. I., 1911, page 481.

MR. E. BURY[2] in investigating the efficiency of coke ovens, made a long series of tests on the effect of temperature on yield, composition, candle and calorific values, and plotted the results obtained at different temperatures by means of curves. From these experiments BURY concludes that in the coke oven the coal is heated at a lower temperature than it is in the gas retort, and that the yield is therefore primarily lower; and that any further addition to the yield takes place at the expense of the illuminants; but even after this has occurred, the total yield is 10 per cent. lower than would have been obtained from the gas retort using the same coal. He also concludes that in the gas retort the heat penetrates rapidly to the center of the charge, with the result that the gas is generated at a higher temperature, leading to a larger yield; and furthermore, that owing to the minimum heating surface presented to the evolved gases, the illuminants escape from the retorts before decomposition has time to take place, thereby maintaining a high yield with a greater heating and lighting efficiency. Apart from the chemical aspect of gas making in the by-product oven and the gas retort, there are some other matters in favor of the latter as a gas-making apparatus; for example, the oven presents far greater opportunity for leakage of gas through the joints of the small bricks of which it is composed than is the case with the jointless retort. Again, the large doors of the oven offer facilities for leakage or else consequent in-draft of air, especially after the hinges begin to work loose. Moreover, the large doors present another drawback in that the coal immediately against them escapes carbonization. This not only means a loss in gas yield, but also the production of a quantity of half-carbonized coke, or black ends. Besides, the coke required for smelting purposes must be absolutely free from volatile constituents, and this tends to keep the coke in the oven for too long a time, resulting in drawing in air and diluting the gas with nitrogen, and also in generating such an amount of heat as to cause a decomposition of the illuminants in the next charge.

The KOPPERS COMPANY, in a paper read by PETERS in 1908, states that in oven operation "the heated wall surface must be covered with coal so that the heat transmitted into the mass may be used for the distillation. It must purposely be avoided to overheat the gas on its way to the off-take. Coke and gas oven chambers are constructed in this manner. The conditions existing in a gas retort are not found in such chamber ovens, for instead of the present primary gases becoming overheated and broken down on the hot surface, these primary products are condensed and redistilled by the advancing heat, giving larger coke yields and an increased ammonia yield, producing a light fluid tar with the elimination of graphite and stopped standpipes. If the walls of a coke oven are heated uniformly and the coal is charged into the oven in such way that the entire heating wall is covered, then there is no possibility of the products of distillation being broken down. If the walls are not uniformly heated, or the coal does not cover the entire heated wall, it is found that the gas becomes inferior, while both tar and ammonia yields fall off. As long as the oven chamber still contains coal in the center of the charge, the temperature of the inside of the oven chamber is comparatively low, and of such a degree that the produced gases do not undergo decomposition."

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[2] Jnl. of Gas Ltg., June 25, 1907, page 982.

DR. BUEB states that, in his opinion, coke ovens cannot replace retort carbonization in gas works for the following reasons:

Coke ovens, operated as such, require more men than do ordinary gas works, and they therefore present additional difficulties in case of strikes; they are, moreover, dependent upon mines and blast-furnaces, and they could not guarantee to deliver gas in the event of either of these being shut down. To replace retort benches with ovens, and to sell the gas produced over and above that required for heating the ovens, would place more than four times the present output of coke on the market, while even now the increased make of coke can be disposed of only by a reduction in price. Attempts have been made to introduce ovens, heated by producer gas, into gas works, and ROTHENBACH tried out ovens of this class at Olten, but the time required for carbonization appears to have been too long, and consequently the cost of fuel was too high. The GERMAN CONTINENTAL GAS COMPANY tried out a system of coke ovens consisting of horizontal carbonizing chambers, so constructed that the separate chambers were divided into two overlying sections; the results were satisfactory, but the system was abandoned because greater advantages were seen in the vertical retort system. The Munich coking chambers are essentially the same in principle as the oridnary coke oven, and there is always a highly superheated space in the upper part of the chamber through which the gas must pass with a consequent dissociation of hydrocarbons, and also with the formation of naphthalene. In this respect the oven is radically different from the vertical retort, in which there is no such decomposition space, and from which the gases may pass away without deterioration.

It seems probable that in course of time the Munich inclined chamber will be built vertically, and experiments have been in progress at Dessau on the enlargement of the present retort, which, if they prove successful, will cause them to represent a vertical oven. It has been said that the introduction of the coke oven would eliminate night work, but DR. BUEB discredits this for the following reasons: (1) Unless a 12- or a 24-hour schedule were adopted, which would not be possible on account of the variation in time required for carbonizing different coals, some of the charging must be done at night; (2) if all the charging were done in the daytime, there must be great fluctuations in the make, necessitating a largely increased plant other than that required for carbonizing; (3) inasmuch as but one stoker per shift suffices for the charging and discharging of the verticals at the OBERSPREE (Berlin) works, and a watchman must be employed under any condition, there would be no economy in eliminating night work; (4) cessation of night work would mean great variation in the quality of the gas; (5) steam could be admitted to the retort only at the close of the carbonizing period, and this would mean a still wider variation in the quality of the gas, owing to all of the water gas being made in the space of a few hours.

J. D. FORREST,[1] in reviewing the operation of the new coke oven plant of the Citizens Gas Company, Indianapolis, states that "in summarizing the advantages of the by-product coke oven, we find that the labor cost per ton of coal carbonized is lower, the quantity, quality and value of coke produced is higher, the fuel requirements for carbonization are more economical, the quantity and quality of the gas are better, and the quantity and quality of by-products likewise are better than the best practice coal gas retorts can obtain. These are certainly advantages which indicate the adaptability of the by-product coke oven to the manufacture of gas for city distribution, *wherever conditions are favorable for the operation of such plants.* However, this qualifying clause which I have used has so much significance in my mind that I must devote the remainder of my paper to showing the reasons why by-product ovens are not well adapted to the purpose of gas manufacture for the great majority of gas companies of the country."

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[1] Indiana Gas Association, March, 1914.

Mr. Forrest explains that the objection to the by-product coke oven occurs first in the difficulty experienced in disposing of the coke, and that it is only when an adequate market for the coke can be secured that the gas company can venture to look to the by-product coke oven for its process of manufacture, and he admits that in the vast majority of cities in this country this difficulty alone is sufficient to prevent the introduction of by-product coke ovens. Another reason given against its adoption is the investment required for the plant, stating that the cost of installation of a by-product coke-oven plant, over a coal-gas retort plant of the same gas capacity, is greater than the cost of a coal-gas retort plant over a water-gas plant of the same gas capacity. Added to this, Forrest states that a further important consideration growing out of the large volume of coke and by-products to be handled is sufficient to cause a decision against by-product coke ovens in a very large number of cases. Again, in addition to the sales end of the coke and by-products business, the problem of coal supply becomes one of magnitude never realized by any gas operator as such. "It is no longer a question of getting a certain tonnage of coal with a certain average volatile matter, such as the manager of a gas plant must meet, but it is a problem of buying a coal which first of all will produce a good coke of the kind for which a market is in sight. Secondarily, this coal must produce gas and by-products, but the enterprise will be a failure unless the quality of coke can be maintained at a high standard. Now the coals available for gas making are much more numerous and more easily located than those available for high-grade metallurgical coke. The coals purchased by a by-product company must be selected with a good deal of care from a somewhat limited area, and must be bought at prices which can be afforded in view of the coke market."

CHAPTER XII

CHAMBER OVENS

Chamber ovens have as their object carbonization in bulk, and they are a development of the by-product coke oven, being built with horizontal, inclined and vertical chambers; the first oven of this character was built in about 1901 at Munich, where an old inclined setting was remodeled to include three chambers in place of the retorts, and at the present time there are somewhat more than 1000 chambers being operated in Europe.

The present Munich system of inclined chamber ovens is being operated at Hanau, Germany, each bench containing three chambers, and it is claimed that they work with but little variation in the daily results. The gas yield amounts to 12,670 cu. ft. per ton of coal, with a calorific value of 580 B.t.u., while the coke used as fuel in the producer amounts to 16.3 per cent. of the weight of coal carbonized. The chambers are charged once in every 24 hours, each charge amounting to 4 tons of coal, half of the battery being charged at 7 in the morning and the other half at 5 in the afternoon.

Before the chambers are charged, the pipes leading to the hydraulic main must be thoroughly cleaned, after which the time required for charging, unless some difficulty is experienced, occupies about ½ hour. There is no night work, and if the production should exceed the demand, a charge is permitted to remain in the chamber for a longer period. A tower, traveling on rails along the front of the bench, contains three electric motors, one for opening the doors of the ovens, another for operating a short length of De Brouwer conveyor contained in the tower, and by which the coke is quenched and deposited in a bunker, and the third for moving the tower. The chambers are 16 ft. long, 9 ft. high, and taper in width from the top down to the bottom, the floor of the chamber being set at an angle of about 45 degrees.

The chambers in use at Munich are 13.1 ft. long, 5.9 ft. high, and 1.64 ft. wide, being inclined so that they may be charged from one end and drawn from the other. The upper end is, however, nearly closed by firebrick, and a special charging dome is provided. The producer is located in front of the setting beneath the floor, while the gas take-off is at the upper end of the chamber. The chambers are charged once every 24 hours, resulting in quite some economy of labor. A summary of the results obtained during a 10 day's run is given below:[1]

Average charge per chamber	6,190 lb.
Make per ton, corrected to 0°C. and 760 mm	10,733 cu. ft.
Gas, mean specific gravity	0.40
C. P. in slit burner, burning 5.3 cu. ft. per hour	10.8
Calorific value, gross	662 B.t.u.'s
Calorific value, net	596 B.t.u.'s
Mean temperature of chambers	1250°C., 2282°F.
Dry coke, on percentage of coal	66.77 per cent.
Consisting of large	61.57 per cent.
Medium and small	3.40 per cent.
Breeze	1.79 per cent.
Condensed products, total	13.44 per cent.
Consisting of tar	6.14 per cent.
Liquor	7.30 per cent.
Coke (dry) in furnace, per cent. of coal weight carbonized	15.32 per cent.
Coke produced, per ton of coal	1,480 lb.
Tar produced, per ton of coal	136 lb.
Ammonia liquor, per ton of coal	161 lb.

Both charging and discharging are accomplished by gravity, but a small pusher is provided for use in case of failure. Four men are employed for five ovens each of three chambers, two on each side of the oven. The heat in the ovens is very uniform, ranging from 1250° to 1300°C. (2282° to 2372°F.), while the waste-gas flue temperature is 350°C. (662°F.).

The development of the Munich inclined chamber oven is reviewed by R. LESSING[2] with the statement that the results of the working of the first large installation were sufficiently convincing to the authorities of the HAMBURG GAS WORKS, to make them decide to erect a battery of ten settings, which has been in operation for some time with perfect satisfaction. The heating economy of the system may be gathered from its fuel consumption of 12 to 15 per cent. of the coal carbonized, with a yield of 12,500 cu. ft. of gas per ton of English coal.

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[1] HERR RIES, Jnl. of Gas Ltg., June 18, 1907, page 810. Gas World, June 22, 1907, page 882.

[2] Jnl. of Gas Ltg., June 22, 1909, page 832.

The works at Moosbach, near Munich, is supplied with benches of three chambers each, the chamber measuring 28.7 ft. on the slope and being placed on an angle of 35 degrees 55 minutes. The coal is elevated to the top of the retort house and distributed in bunkers, the latter also serving as measuring chambers. The oven is charged by dropping the contents of a bunker into a traveling chute inserted into the charging door, while discharging is effected by raising the hinged bottom door by means of a small crane, the mass of coke then sliding into the quenching tower.

The works at Bochum carbonize 56 tons of coal per day, and yield over 11,000 cu. ft. of 550 B.t.u. net gas per ton, while at Vienna, where 120 tons are carbonized per day, the yield is from 10,500 to 12,000 cu. ft. of gas per ton, depending upon the kind of coal used. It is claimed that at the latter plant the make per man per shift of from 8 to 10 hours is 175,000 cu. ft.

The advantages claimed for the Munich system are:

Long period charges, with economy of labor and elimination of night work; a high make of gas of good calorific value; little deposit of carbon in retorts or ascension pipes; low fuel consumption; coke of good quality and size; thin tar which contains but little free carbon; small heating surface per ton carbonized, and less chance for the decomposition of illuminants and filtration of flue gases; great capacity for a small ground space; simple and comfortable operation for the attendants; the life of the plant should be considerably in excess of that of the ordinary retort plant.

The disadvantages are:

Low candle power of gas and high cost of installation, it being claimed, however, that the latter is soon made up by the greater economy in working and smaller capital charges on account of higher durability.

LESSING states that the quality of the gas made in a chamber oven gives rise to quite some theoretical consideration. An examination of the curves showing the yield of gas, tar and liquor, calorific value, specific gravity and candle power of the gas throughout the carbonizing period indicate that the candle power is the only one that shows any considerable variation, starting with 27 candles, falling abruptly in the first 4 hours to about 11, and then more gradually to zero at the end of the 24 hours, and he offers the following explanation: Carbonization sets in at the walls of the chamber, allowing the gases evolved to pass freely through the entire mass of the coal, which is still cool. This period corresponds with that of the initial high candle power. Besides, the great mass of the coal must exercise a cooling effect on the chamber walls for a considerable period; the rich hydrocarbons, therefore, pass undecomposed along the roof of the chamber. As soon as the heat has sufficiently penetrated the outer layers, so as to be able to soften the undecomposed coal, an impenetrable, or semi-permeable zone is formed, forcing the gas along the walls to the crown of the chamber. By contact with the hot coke and walls the gas will suffer a degradation with incidental deposition of carbon. This corresponds with the rapidly decreasing illuminating power. This theory finds a certain support in the fact that the cake of coke stands away from the walls of the oven by sometimes as much as an inch. Further, it is split in the center by a vertical plane along its longitudinal axis. The first observation indicates that the coal, while softening, has a tendency to consolidation by filling the intervening spaces, and is assisted in this by the pressure put on the gas by the condensation of the heavier hydrocarbons. This slight pressure may be sufficient to affect this tendency in the direction towards the center, thus making room for the gas to escape along the side walls.

M. C. SISSINGH[1] reports a test made at Rotterdam to determine whether it was possible to eliminate night shifts without too great a variation in the quality of the gas. This plant consisted of four KLÖNNE chambers, and they were charged at 6 and 8.45 A.M., and at 1.30 and 4.15 P.M. The gross heating value, during the test, rose from 540 B.t.u. at 6 A.M. to 719 at about 5 P.M., and then fell to the original figure at 6 o'clock the next morning. The specific gravity of the gas starting at about 0.35, rose to 0.42 at 5 o'clock, then fell to a minimum of 0.32 at midnight, after which it rose to about the figure at which it started. The candle power, measured by a Metropolitan No. 2 burner, rose irregularly from 14 at 6 o'clock A.M. to 21.1 at 6 P.M., and then fell uniformly to about the original figure. In general, the yield was 11,122 cu. ft. of gas of 16.2 candle, 602 B.t.u., and 0.37 gravity, the gas of best quality being obtained about 6 P.M., the period of maximum demand.

In reviewing this statement, the editor of the Gas World[2] calls attention to the wide fluctuation in the quality of the gas, and expresses the opinion that ordinary retorts which may be charged at intervals of from 6 to 12 hours, as desired, are preferable for gas works.

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[1] Gas World, Oct. 9, 1909, page 434.

[2] Oct. 16, 1909, page 451.

The coking chambers at Weimar, Germany, are 13 ft. long, 5 ft. 7 in. high, and taper from 18 in. at the charging end to 20 in. wide at the discharging end. The coking period is from four to five times as long as with retort settings; consequently these chambers are not as convenient as retorts in meeting the fluctuating demand for gas, and they require more care to ensure the production of a gas possessing uniform quality. The yield of gas is, however, higher than with retort settings occupying the same ground space, the coke has a higher value, there is less trouble due to the formation of naphthalene, the chambers require less labor, and it is also claimed that the cost of maintenance is considerably less. 10,400 cu. ft. of gas are produced per ton of coal (2000 lb.) and the calorific value of the gas is 570 B.t.u. gross, while the resultant coke is said to be very dense and hard, weighing from 31 to 34 lb. per cubic foot, the total coke yield being 70 per cent. of the weight of coal carbonized. The coke used as fuel in the producers amounts to 18.3 per cent. of the coal carbonized, but it is claimed that in newer settings with larger regenerators, this fuel consumption has been reduced to 15.5 per cent. The tar produced amounts to 3.68 per cent. of the weight of coal carbonized, which is less than with retort settings, but it is claimed that the tar is free of water; the ammonia yield amounts to 0.234 per cent., or 4.68 lb. per ton of coal. It is further claimed that only 5512 lb. of graphite were removed after making 159 million cu. ft. of gas, or only one-sixth as much graphite as with retort settings, but the cost of the installation was about 25 per cent. higher than the cost of retort settings of the same capacity.

Vienna possesses two chamber oven installations on the KOPPERS system, the plant at Simmering consisting of 46 ovens, with a daily capacity of 50 million cu. ft., while one at Leopoldau comprises 72 ovens with a capacity of 75 million cu. ft. of gas, the ovens being heated, as explained in a previous chapter, by an independent producer plant. A test was run on the first eighteen of these ovens on September 12 to 18, 1911, with the following results:

Average length of charge	**23 hours 44 minutes**
Average time of charging and discharging.	**9 minutes**

The coal used gave the following analysis:

	Per cent.
Moisture	**1.36**
Ash	**10.13**
Combustible	**88.51**

During the test run 1170 tons of coal were carbonized, yielding

Gas, corrected to 60°F., 30-in. bar., 8-in. holder pressure	**13,567,000 cu. ft.**
Yield per ton of 2240 lb	**11,600 cu. ft.**
Calorific value of gas	**614 B.t.u.**

Analysis of gas:

	Per cent.
Carbon dioxide	**2.6**
Illuminants	**2.9**
Oxygen	**0.3**
Carbon monoxide	**9.1**
Hydrogen	**53.3**
Methane	**28.2**
Nitrogen	**3.6**

Coke used in producers:

	Tons
Size from 0 in. to ½ in	14
Size from ½ in. to 1 in	164
Total	178

Analysis of coke:

		Per cent.
0 to ½ in.:	Ash	16.89
	Moisture	18.67
	Combustible	63.54
½ to 1 in.:	Ash	16.14
	Moisture	17.97
	Combustible	65.89

E. A. Franks[1] reports on a chamber oven plant in which each chamber has an average capacity of 2 tons 15 cwt. of coal, one furnace being provided for each three chambers, or two furnaces for the entire plant. The floor of the chamber has a slope of 45 degrees, the roof sloping at 32 degrees. The chamber is charged by gravity through a door at the top of the rear end, the coal being brought to the charging point in a traveling hopper operated by one man. The coke, as it is discharged from the chamber, falls into a subway located in front of the battery, and it is loaded from there by hand into barrows and wheeled to a steam lift, the latter raising it to the ground level. Each furnace is 4 ft. square and has a depth of 13 ft., the gas outlet being located at a point 4 ft. 6 in. above the fire bars at the rear of the furnace, this depth permitting of a continuous run without filling for 16 hours, but requires occasional pricking of the fuel.

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[1] Gas World, May 31, 1913, page 731.

The results of 30 weeks' operation were:

Coal carbonized	2637 tons, 1 cwt.
Gas made	28,346,800 cu. ft.
Coke made	1438 tons, 1 cwt.
Coke sold	1155 tons, 16 cwt.
Tar produced	11,551 gal.
Gas per ton of coal	10,760 cu. ft.
Coke sold per ton of coal carbonized	9.3 cwt.
Tar produced per ton of coal	10.153 gal.

An analysis of the gas gave:

	Per cent.
Carbon dioxide	2.6 to 3.4
Illuminants	2.6 to 3.4
Oxygen	Trace
Carbon monoxide	6.1 to 7.3
Methane	25.5 to 26.5
Hydrogen	57.0 to 58.0
Nitrogen, by difference	3.0 to 4.0
Mean illuminating value	14.1 candles

CHAPTER XIII

LOW TEMPERATURE CARBONIZATION

Low temperature carbonization has received but little attention in the United States, but quite some successful work has been done in this direction in Europe, especially in England. PARR and OLIN[1] have directed their attention to this subject since 1907, and their studies developed three lines of industrial interest.

First: Their attention has been drawn to the possibility of developing a fuel of good texture, which shall be smokeless, and which shall be suitable for domestic as well as for general industrial use, and they assert that the by-products obtained while making such a fuel promise to be of special value. These products consist of gas of a high illuminating and calorific power; of ammonia, the quantity of which will of course be less than that obtained with higher temperatures; and tar, the latter being composed almost entirely of oils, containing a minimum quantity of pitch and free carbon.

Second: A possible method is suggested for the production of a producer gas which would be free from present difficulties attending the use of bituminous coal, and which would convert a much higher percentage of the fuel into gaseous form.

Third: Quite interesting possibilities in the production of coke are opened up by the application of this process; this also holds good for the production of briquettes or other forms of fuel manufactured into a dense and stable form of such consistency as will meet the requirements of shipping, storage, foundry, or other industrial uses.

The apparatus employed by PARR and OLIN is illustrated in Figs. 100 and 101. High pressure steam is admitted from the main at (*A*) into the coil (*BB*), the latter being made of a 5/8-in. pipe, 11 ft. long, fitted with two return elbows, the steam being heated by means of a 26-burner combustion furnace (*CC*) located under the coil. The retort (*D*), 8 in. in diameter and 18 in. long, was supplied with the cylindrical sheet-iron cylinder, 6 in. in diameter, Fig. 101, perforated with small holes as shown, for the purpose of containing the coal charge; this shell, having a smaller diameter than the retort proper, had a surrounding space of about 1 in., and thus permitted a free distribution of the heat. The retort proper was fitted with a head (*J*), attached to the body by means of tap bolts, the joint being sealed with an asbestos gasket. From the retort, the distillates were conducted by means of a pipe to the condenser (*E*), which latter was in turn connected to a large wash-bottle (*F*), where the oils and tars were collected while the gases passed on to the gasometer (*G*). By means of a stuffing-box (*S*), a HOSKINS nickel-nichrome thermocouple was inserted into the retort, the couple being connected to a millivoltmeter at (*K*), for the purpose of measuring the temperature of the retort contents. A battery of burners was placed directly under the retort for the purpose of providing additional heat, the latter being retained by means of an oven lined with asbestos, this oven entirely surrounding both the retort and the furnace.

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[1] Un. of Illinois, Bul. 60. The Coking of Coal at Low Temperatures, by S. W. PARR and H. L. OLIN.

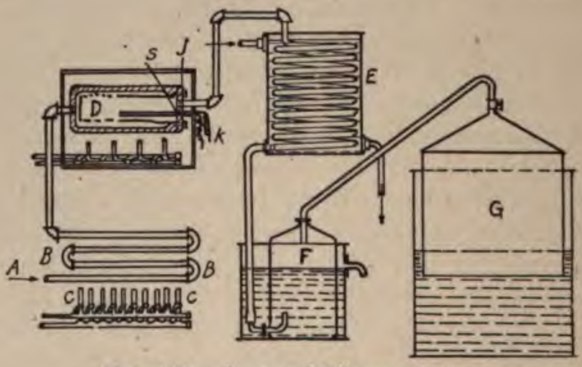


FIG. 100.—Parr and Olin apparatus.

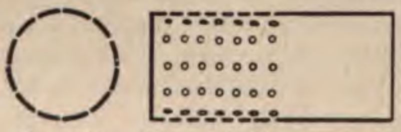


FIG. 101.—Parr and Olin apparatus.

Heat was transmitted into the charge of coal through the medium of superheated steam, which readily lent itself to an even distribution of heat throughout the coal. PARR and OLIN'S studies were made with the purpose of determining the coking powers of various coals, irrespective of their relative commercial values, and therefore many of the samples used were cleaner than the general run-of-mine coal would have been, the low ash and sulphur contents noted in Table LXXXIII resulting from the exclusion of pyrites.

TABLE LXXXIII.—COMPOSITION OF COAL USED BY PARR AND OLIN

Mines, Counties Illinois	Moisture	Ash	Volatile matter	Fixed carbon	Sulphur	B.t.u.
Vermillion	8.80	8.72	43.05	39.43	2.88	12,673
Franklin	6.84	7.38	37.96	47.82	1.33	12,770
Saline	3.93	5.80	37.86	52.41	1.54	13,593
Macon	8.70	12.12	39.30	40.88	2.30	11,417
Perry	7.19	10.05	35.42	47.34	0.80	12,153
Williamson	5.30	8.55	36.50	49.65	2.77	12,640

Sufficient coal, from 2500 to 3000 grams, for one charge was crushed at one time, the size of the pieces in the first experiments ranging from buckwheat size to ¼ in., the dust being removed by means of a sieve.

After the retort was charged, steam was admitted for the purpose of blowing the system until all air had been expelled, after which the combustion furnace was started, followed by the burners under the retort, the results from average working conditions being given in Table LXXXIV.

TABLE LXXXIV.—TEST CONDITIONS. PARR AND OLIN

Run No.	3	4	5	6	7
Weight of coal, grams	4800	5351	2195	3498	3398
Weight of residue, grams	4030	4112	1895	2810	2895
Maximum temperature, C.°	475	515	450	410	430
Maximum temperature, F.°	887	959	842	770	806
Ratio of coke, per cent	84	76.8	86.3	80.3	85.2

The distribution of products, using coal from the Electric Mine, Danville, Ill., is given below:

Temperature (average)	450°C. (842°F.)
Time of distillation	5 hours.
Volatile matter in original coal, not including moisture	43.00 per cent.
Volatile matter in coke residue	27.95 per cent.
Volatile matter in coke residue referred to original coal	22.01 per cent.

Loss in weight of original coal, volatile matter only, not including moisture	20.28 per cent.
Total volatile matter derived as above, not including moisture	42.29 per cent.
Total material, removed by distillation, including moisture	29.10 per cent.

The gas produced from the Danville coal analyzed:

H_2S	3.2
CO_2	5.7
Illuminants	8.3
CO	5.2
H_2	5.0
C_2H_6	14.4
CH_4	51.4
N_2	5.7
B.t.u.	1032

while the yield of products for different periods of heating is given in Table LXXXV.

TABLE LXXXV.—YIELD OF PRODUCTS. PARR AND OLIN

Time of heating	3 hours	6 hours
Coal, grams	3000.0	4000.0
Coke, grams	2327.0	2902.0
Coke, per cent	77.50	72.50
Weight of tar, grams	238.5	316.0
Tar, per cent	7.93	7.90
Weight of total water, grams	208.5	348.4
Free moisture, per cent	3.38 or 6.93	3.00 or 8.71
Water of constitution, per cent	3.55	5.71
Volume of gas at 760 mm. and 0°, liters	87.0	134.7
Calculated to cu. ft. per lb. of coal, cu. ft	0.46	0.54

This gas has a relatively high heat value, the computed value being 1024 B.t.u., while that given by the calorimeter is 1032 B.t.u. per cubic foot. Comparing this with the usual gas at 600 B.t.u. per cubic foot, this gas has a calorific value about 70 per cent. greater, or 1 cu. ft. of this gas at 1024 B.t.u. is equal to approximately 1.7 cu. ft. at 600 B.t.u.

The distillation of Illinois coals at temperatures between 450° and 500°C. (860° and 932°F.) produces a gas possessing a heating value exceeding 1000 B.t.u. per cubic foot, but the yield approximates only ½ cu. ft. per pound of coal which, at the heat value present, would represent a yield of 1 cu. ft. of a gas having a heat value of about 500 B.t.u. per cubic foot. The ammonia yield also is low, being only about 3 lb. of ammonium sulphate per ton of coal carbonized, while decomposition at this temperature extends to the oxygen compounds, which are, in general, carried off and appear in the condensate instead of in the gaseous product.

As stated above, the amount of tar recovered from the distillation approximates 50 per cent. of the yield of volatile matter, and represents almost 8 per cent. by weight of the original coal. A distillation test of this tar gave the following fractions:

Amount of tar (exclusive of water carried over)			375.0 grams = 5775 grains
Light oil,	(20°–100°C.)	68°–212°F.	39.1 grams = 10.5 per cent.
Fraction (*b*),	(100°–200°C.)	212°–392°F.	109.1 grams = 29.1 per cent.
Fraction (*c*),	(200°–240°C.)	392°–464°F.	111.8 grams = 29.8 per cent.
Fraction (*d*),	(240°–275°C.)	464°–527°F.	20.6 grams = 5.5 per cent.
Coke residue			80.0 grams = 21.3 per cent.

About 75 per cent. of the material classed as tar is, from the above, really oils of different specific gravities, and therefore of much greater monetary value than is pitch proper, and this latter product is much smaller in amount than is usually produced with high temperature carbonization.

The coke yield, under average conditions is approximately from 75 per cent. to 80 per cent., but this factor will vary greatly with the amount of ash in the original coal, as well as on the temperature of distillation. These items of variation are plainly shown in Table LXXXVI, where material of widely varying composition was used.

TABLE LXXXVI.—COMPOSITION OF COKE RESIDUES

Constituent	Vermillion coal	Franklin Co.	Saline Co.
Moisture	0.34	0.40	0.28
Ash	11.15	9.28	6.97
Volatile matter	27.61	26.60	23.50
Fixed carbon	59.90	63.72	69.23
Sulphur	2.58	1.21	1.20
B.t.u.	12,892	13,446	13,746

YIELD OF COKE FROM VARIOUS COALS REFERRED TO ORIGINAL COAL—DRY BASIS

Ash	9.56	7.92	6.04
Volatile expelled	25.48	18.00	19.12
Residue coke	78.10	84.72	84.86

The coke material obtained by this low-temperature process varies in character with the character of the coal employed as well as to the amount of pressure employed during the period of carbonization. For example, Williamson County coal produces a coke of much finer texture and less porosity than does Vermilion County coal, and, in order to determine the reason for this greater porosity, or in order to determine the conditions which would modify it, PARR and OLIN carried out a test with the coal sample under pressure in the apparatus illustrated in Fig. 102. This apparatus consisted of an iron cylinder (A) 8 in. by 4 in., each end being closed with a screw cap, (B) and (B'), after the coal charge had been inserted. The movable piston (C), actuated by the rod (D) was pressed against the coal by screwing down the nuts (E), the cylinder being perforated as shown to permit the escape of gases. This apparatus was placed in the retort shown in Fig. 100, and heat was applied as before.

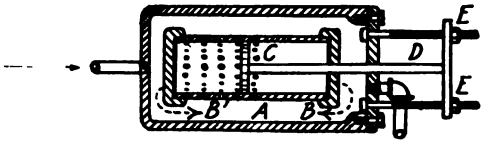


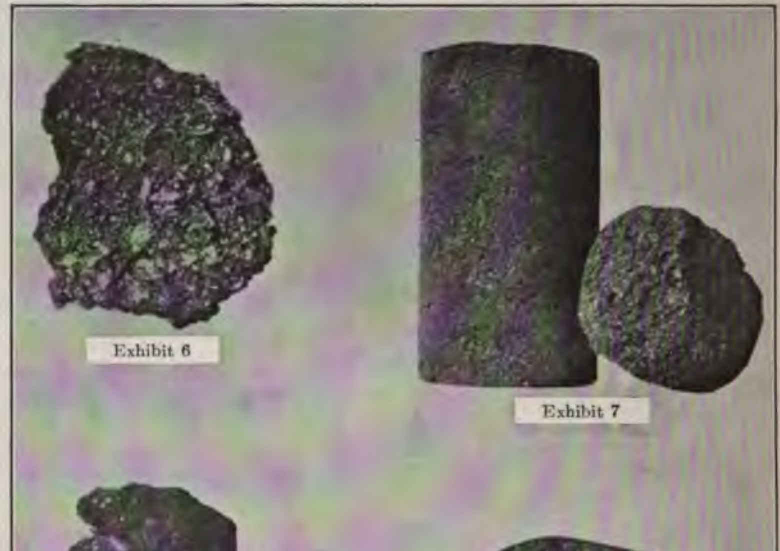
FIG. 102.—Parr-Olin pressure retort.

Exhibit 1 shows the results obtained when pressure was applied gradually during the entire period of distillation, the outer portions of the sample passing through the temporary state of fusion, soon hardening and forming a wall which resists external pressure, thus making the inner core extremely porous. If sufficient pressure is applied the outer portion will fracture, and the residue will come out broken up into small pieces. Coal from Perry County was used in producing exhibit 1, the specific gravity of the outer portions of this coke mass being 0.733 as against 0.652 when coked without pressure. During the next test the charge was rammed into the cylinder and the piston was screwed up tightly, no further pressure being applied after heating had begun. The resulting column cohered well and it exhibited the same increase in specific gravity as the one mentioned above.

An interesting feature of the product due to this process is the complete fusion of the mass under proper conditions, the individual particles of coal of buckwheat size having completely lost

Exhibits 1 to 5.

PLATE II.



Exhibits 6 to 10.

their identity, while the resulting homogeneous mass shows no lines of demarcation from the original pieces of coal, but the texture is finer or closer in some cases than in others, exhibits 2 and 3, made from southern Illinois coals, very well illustrating these points. Exhibit 4, made from Vermilion County coal, represents a somewhat coarser texture. Exhibit 2, made from Saline County coal, had a crushing strength of 750 lb. per square inch, while exhibit 3, made from Franklin County coal, crushed at 900 lb., and exhibit 4, on account of its coarse cellular structure, showed but little rigidity, and it broke down at a pressure of 300 lb. Exhibit 5 shows a mass of material which exuded from a sample of Vermilion County coal which was distilled without pressure, and exhibit 6 is an illustration showing the conditions which exist where oxidation has been allowed to take place.

Exhibits 7, 9, and 10 show the results of briquetting tests made with different materials.

These experiments developed the following fundamental facts:

1. "The formation of coke depends upon the presence of certain constituents having a melting point which is lower than the temperature at which decomposition or carbonization takes place."

2. "Oxidation of these compounds may easily take place and the greatest coking effect is obtained where the opportunity for the minimum amount of oxidation has occurred. The condition prescribed, therefore, is that there shall be the least possible exposure to oxidation either before or during the process of carbonization."

3. "Coals containing an excessive quantity of the coking substance produce a light porous coke. The texture of the product may be modified by use of pressure and by close packing of the charge and especially by mixing with material which has already passed through the coking process. Such a mixture provides the physical conditions whereby the gases formed may readily pass out of the mass without carrying along the cementing substances."

4. "By use of temperatures between 400° and 500°C. (752°–932°F.) all of the resulting products are of a type distinctly different from those obtained by the usual high-temperature procedure."

Parr and Olin continued their experiments with a coking retort capable of distilling 100 lb. of coal, and they deduced the following very interesting summary therefrom.

SUMMARY

1. "Coke of good density and hardness may be made by mixtures of semi-coke and raw coal if both are finely divided and evenly mixed. A

variation is noticeable in the quantity of such non-coking material which may be incorporated with different coals. For example, fresh coal from Vermilion County will carry such an addition of 100 per cent. of its weight to advantage. Coals from Saline and Williamson Counties give coke of the highest density when mixed in the proportion of 50 per cent. of their weight with semi-coke."

2. "The coke resulting from the low-temperature process has from 18 to 22 per cent. of volatile matter remaining, but since it has been heated above 400°C. (752°F.) there should be none of the tar constituents remaining. The most convincing test on this point, as also the best method of arriving at a conclusion as to its adaptability for such work, was to try out the material in a suction gas producer. The results indicated that no clogging effect whatever results, thus showing the absence of tar bodies. The physical operation of the producer, as well as the grade of the gas produced, was fully equal if not superior to the performance of the outfit when anthracite was used."

3. "The semi-coke has such an amount of volatile matter remaining, together with the right degree of coherence as to make it especially well adapted to household use. It is clean to handle, free from dust, and burns without smoke or the formation of soot. Especially to be noted in this connection, is its ability to retain a fire without undue attention as to drafts, etc."

4. "The average specific gravity of the tar is 1.069. It is rich in low boiling distillate passing over at 210°C. (410°F.). This product averages 18 per cent. of the total. The pitch residue amounts to approximately 30 per cent. and is remarkably free from precipitated carbon."

5. "The adaptability of the tar for wood preservation processes seems to be indicated by the high percentage of tar acids. These constituents make up from 28 to 30 per cent. of the crude material. The larger part, about 22 per cent., is found in the second distillate 210° to 325°C. (410°–615°F.), only about 7 per cent. coming over below 210°C."

6. "Approximately 10 per cent. of the crude tar is found to be low boiling distillate free from the tar acids and suitable for use in internal combustion engines."

7. "Naphthalene is absent. The free carbon in the crude tar is less than 2 per cent., and the residual product after the light distillate and heavy oils are removed, would be classed as hard pitch."

8. "A principal feature results from this study of these various substances, namely, that all three of the general divisions of coke, tar, and gas have specific properties of an especially valuable sort, which would indicate that the process of coking at low temperatures could be established successfully on a commercial basis."

G. E. Foxwell[1] states that as worked at present, low-tempera-

ture carbonization systems are all designed with a view to producing a free-burning, smokeless coke, and a high yield of oils. Gas works coke contains anything up to 2 per cent. of volatile matter as a rule, and it will burn only with a strong draught, for which reason it does not always find favor for household purposes, but if the coke should contain from 5 to 6 per cent. of volatile matter, it would burn freely without the production of smoke. Low-temperature carbonization systems therefore aim at retaining between 5 and 10 per cent. of the volatile matter of the coal in the coke, at the same time securing a high yield of oils, these oils being secured partly from the tar which is condensed and extracted in some form of tar extractor, and partly by stripping the gas, the latter process being analogous to the removal of benzol from coke-oven gas. Some systems require that distillation be carried out under a heavy vacuum, between 15 and 20 in. of mercury, while in others the retorts are operated under pressure, or gas is passed through the carbonizing mass to sweep away the products of distillation. The various systems also operate under various temperatures, the usual range being between 450° and 650°C. (842° and 1202°F.); the tar produced at the lower temperature is of a wholly paraffinoid character, strongly resembling petroleum in many respects, but as the temperature is increased towards 600°C. (1112°F.) benzene bodies appear and an intermediate tar results. The amount of ammonium sulphate produced varies between 20 and 28 lb. per ton of coal carbonized.

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[1] London Jnl. of Gas Ltg., July 14, 1914, page 104.

The retorts used are usually inclined or vertical, the retorts, made of iron, in one system being placed at an angle of 15 degrees; the coal is conveyed from a hopper at the lower end through the retorts by means of screw conveyors. The retorts are heated by a portion of the gas evolved, this gas being distributed by means of perforated iron pipes situated beneath the retorts, while the length of time to which the coal is subjected to the distilling process is regulated by the speed of the conveyors. FOXWELL cautions that one of the principal points to be remembered in connection with low-temperature carbonization is that the layer of coal should be as thin as possible, and in one of the best-known systems this condition has been especially taken care of in the design of the retort, which is a vertical one, consisting of two concentric annular rings, each 2.5 to 3 in. in diameter, and a small central ring. The coal is placed in the two outer rings, a portion of the hot gases of distillation passing up the center ring, thus helping to heat the interior while the retorts are also being heated

the retorts being operated under a vacuum of [illegible].

[illegible] gases have been evolved by the process of distillation, [illegible] to the condensing plant, and the heavy hydrocarbons are there first removed by means of a P. and A. tar [illegible]. The calorific value of the gas at this point amounts to [illegible] B.t.u. per cubic foot. After passing the exhauster, [illegible] subjected to a pressure of from 50 to 60 lb. per square [illegible]. The pressure is then suddenly released, causing a quantity [illegible] light oils to condense, and thus lowering the calorific [illegible] the gas to 750 B.t.u., after which the remaining light [illegible] ammonia are removed in the usual manner by scrubbing. [illegible] gas, which now has a heating value of about 300 B.t.u. per [illegible] foot, is returned to heat the retorts, the amount of gas [illegible] in this manner being from 5000 to 7000 cu. ft. per ton [illegible] carbonized.

Foxwell states that low-temperature coke has been found [illegible] for use in water-gas plants but, owing to its low specific gravity, it is found that the blast pressure must be reduced, this reduction in pressure being counteracted by the inflammable character of the coke. The rich gas coming from the low-temperature retorts can be mixed with the blue water gas, thus considerably raising its heating power. **Foxwell** calls attention to the fact that since the candle-power standard is being rapidly superseded by one based on calorific values, which will probably be the standard of the future, all figures given, and estimates made by him are based on calorific value only, and he gives the volume of gas and calorific value as follows:

Gas evolved from low-temperature distillation of coal = 5000 cu. ft. with a calorific value of 1000 B.t.u. per cubic foot.....	5,000,000 B.t.u.
Gas obtained by gasifying low-temperature coke in a water-gas plant (= 70,000 cu. ft. per ton of fuel) assuming 14 cwt. of coke per ton of coal carbonized as being gasified in this plant = 49,000 cu. ft., with a calorific value of 320 B.t.u. per cubic foot.	15,680,000 B.t.u.
Total, 54,000 cu. ft. of gas..........	20,680,000 B.t.u.

and the combination gas would have a calorific value of 383 B.t.u. per cubic foot.

Foxwell assumes that in ordinary gas-works practice there is a yield of 60 per cent. of coke and 12,000 cu. ft. of gas per ton of coal carbonized, or that 1 cwt. of coke is produced for every 1000 cu. ft. of gas made, and that the disposal of this quantity of coke will give the gas manager some trouble. He then assumes that 0.75 cwt. of coke will be made for every 1000 cu. ft. of gas produced in the low-temperature process, and thus secures the following figures:

Total gas produced from 1 ton of coal (as given above) = 54,000 cu. ft.	20,680,000 B.t.u.
One ton of coal produces about 14 cwt. of smokeless fuel; allowing 0.75 cwt. of coke to each 1000 cu. ft. of total gas output, 3.83 tons of coal would be required. Gas produced from low temperature distillation of 3.83 tons of coal = approx. 19,150 cu. ft. of gas of 1000 B.t.u.	19,150,000 B.t.u.
Total, 73,150 cu. ft. of gas	39,830,000 B.t.u.

and this is equivalent to a gas having a calorific value of 544.5 B.t.u. per cubic foot. The gas is of good quality, and its heating value can be varied at will by increasing or decreasing the amount of low-temperature rich gas mixed with the water gas.

In order to arrive at a reasonable estimate of the finances involved by such a scheme, Foxwell took a set of figures issued by a firm manufacturing low-temperature equipment, and deducted 25 per cent. from all yields except coke and ammonia, assuming that this reduction would represent the difference between test conditions and ordinary working. He states that the total yield of oil from bituminous coal ranges between 16 and 22 gal. per ton, and taking the lower figure of 16 gal., the yields obtained on fractionation would be, per ton of coal carbonized, about:

Light oils	1.0 gal.
Middle oils	4.3 gal.
Lubricating oils	3.0 gal.
Pitch	70 lb.

while the other by-products obtained from the distillation of a ton of coal are:

Coke	14 cwt. (70 per cent.)
Ammonium sulphate	24 lb.

In the case of a works producing 3,000,000 cu. ft. of gas a day, and with a gas yield of 12,000 cu. ft. per ton of coal carbonized, the coal used per day would amount to 250 tons. For every ton

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[1] London Jnl. of Gas Ltg., July 14, 1914, page 104.

The retorts used are usually inclined or vertical, the retorts, made of iron, in one system being placed at an angle of 15 degrees; the coal is conveyed from a hopper at the lower end through the retorts by means of screw conveyors. The retorts are heated by a portion of the gas evolved, this gas being distributed by means of perforated iron pipes situated beneath the retorts, while the length of time to which the coal is subjected to the distilling process is regulated by the speed of the conveyors. FOXWELL cautions that one of the principal points to be remembered in connection with low-temperature carbonization is that the layer of coal should be as thin as possible, and in one of the best-known systems this condition has been especially taken care of in the design of the retort, which is a vertical one, consisting of two concentric annular rings, each 2.5 to 3 in. in diameter, and a small central ring. The coal is placed in the two outer rings, a portion of the hot gases of distillation passing up the center ring, thus helping to heat the interior while the retorts are also being heated

variation is noticeable in the quantity of such non-coking material which may be incorporated with different coals. For example, fresh coal from Vermilion County will carry such an addition of 100 per cent. of its weight to advantage. Coals from Saline and Williamson Counties give coke of the highest density when mixed in the proportion of 50 per cent. of their weight with semi-coke."

2. "The coke resulting from the low-temperature process has from 18 to 22 per cent. of volatile matter remaining, but since it has been heated above 400°C. (752°F.) there should be none of the tar constituents remaining. The most convincing test on this point, as also the best method of arriving at a conclusion as to its adaptability for such work, was to try out the material in a suction gas producer. The results indicated that no clogging effect whatever results, thus showing the absence of tar bodies. The physical operation of the producer, as well as the grade of the gas produced, was fully equal if not superior to the performance of the outfit when anthracite was used."

3. "The semi-coke has such an amount of volatile matter remaining, together with the right degree of coherence as to make it especially well adapted to household use. It is clean to handle, free from dust, and burns without smoke or the formation of soot. Especially to be noted in this connection, is its ability to retain a fire without undue attention as to drafts, etc."

4. "The average specific gravity of the tar is 1.069. It is rich in low boiling distillate passing over at 210°C. (410°F.). This product averages 18 per cent. of the total. The pitch residue amounts to approximately 30 per cent. and is remarkably free from precipitated carbon."

5. "The adaptability of the tar for wood preservation processes seems to be indicated by the high percentage of tar acids. These constituents make up from 28 to 30 per cent. of the crude material. The larger part, about 22 per cent., is found in the second distillate 210° to 325°C. (410°–615°F.), only about 7 per cent. coming over below 210°C."

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and the calorific value per pound of gas will be

$$0.2456 \times 1 \times 950 = 233.32 \text{ B.t.u.}$$

The volume of the above flue gas at 950°F. (510°C.) will be:

$$\begin{array}{lll} \text{Fuel: } 388{,}876 + \{(950 - \ \ 60)(0.00206)\}\ 388{,}876 & = & 1{,}101{,}841 \text{ cu. ft.} \\ H_2O\text{: } \ 40{,}213 + \{(950 - 212)(0.00206)\}\ \ 40{,}213 & = & \underline{\ \ 101{,}337 \text{ cu. ft.}} \\ \qquad\text{Total volume} & = & 1{,}203{,}178 \text{ cu. ft.} \end{array}$$

Waste-heat Boiler.—The waste heat boiler is of the Babcock-Wilcox type, arranged as shown in Fig. 103, and it is to produce steam of 100 lb. pressure, corresponding to 340°F. in temperature. Allowing a maximum passage of 4000 lb. of flue gas per hour through the boiler passes, and a minimum of 1 sq. ft. of heating surface per 10 lb. of flue gas, we have a temperature factor, determined experimentally, of 0.132, and the temperature of the gases leaving the boiler will be

$$340 + 0.132(950 - 340) = 420°\text{F.};$$

consequently the drop in temperature while passing through the boiler will be

$$950 - 420 = 530°\text{F.}$$

The boiler, as designed, has the following gas passage:

$$\begin{array}{ll} \text{Width of pass} & = 53 \text{ in.} \\ \text{Tube space, } 7 \times 4 & = \underline{28 \text{ in.}} \\ \text{Net width} & = 25 \text{ in.} = 2.07 \text{ ft.,} \end{array}$$

while the length of the pass is 4 ft. 6 in.; the area of the pass will then be

$$2.07 \times 4.5 = 9.315 \text{ sq. ft.}$$

and the weight of gas passing through per hour will be

$$\frac{35{,}575.6}{9.315} = 3820 \text{ lb.,}$$

which is within the maximum of 4000 lb. allowed.

Allowing, as per the above, 1 sq. ft. of heating surface per 10 lb. of gas, we would require

$$\frac{35{,}575.6}{10} = 3575 \text{ sq. ft.,}$$

while the boiler, as designed, has 3930 sq. ft. of heating surface.

The horse power of the boiler will be given by

$$\text{Hp.} = \frac{W(T - t)S}{33{,}300} \tag{54}$$

in which W = the weight of gas per hour = 35,575.6 lb.
$T - t$ = drop in temperature = 530°F.
S = specific heat of the gas = 0.246;

consequently the horse power under the above conditions will be

$$\frac{35{,}575.6 \times 530 \times 0.246}{33{,}300} = 139$$

The total cost of a recent waste-heat boiler installation, consisting of six water-tube boilers, each of 360 hp., including boilers, hydraulic operated gas valves, waste-heat take-off, stacks, blow-off tanks, piping, operating floor (the boilers were elevated), boiler supports, addition to building, and boiler accessories, amounted to $120,863.00, the annual repairs allowed amounting to $4000.00.

The cost of steam production was made up as follows:

Water	$3,037.00
Labor, 300 days, 3 men at $3.25	2,925.00
Labor, cleaning, 300 days at $3.25	975.00
Repairs, per year	4,000.00
Interest, 6 per cent. on cost of plant	7,252.00
Depreciation, 10 per cent. on cost of plant	12,086.00
Total per year	$330,275.00

At 289,656,000 lb. of steam per year, the cost of steam, per 1000 lb. with waste-heat boilers therefore amounted to

$$\frac{\$30{,}275.00}{289{,}656} = 10.4 \text{ cts.}$$

The cost of a recently built hand-fired boiler plant, comprising five water-tube boilers, each of 440 hp., including boilers, foundations, building, chimney, piping, and accessories, amounted to $85,453.00, the annual repairs allowed amounting to $2000.00.

The cost of producing steam with this plant was:

Water	$4,920.00
Coal	64,350.00
Labor, 300 days, 8 men at $3.25 (12-hour shifts)	7,800.00
Labor, cleaning, 300 days at $3.25	975.00
Repairs per year	2,000.00
Interest, 6 per cent. on cost of plant	5,127.00
Depreciation, 10 per cent. on cost of plant	8,545.00
Total per year	$93,717.00

.000 lb. of steam per year, the cost of hand-fired boiler, will therefore be:

$$\frac{5.717.00}{175.200} = 19.7 \text{ cts.},$$

ected by the use of waste-heat boilers under therefore amounts to $19.7 - 10.4 = 9.3$ cts. per made.

al in the latter example was \$3.25 per ton in while the cost of water in both cases was 65 cts.

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CHAPTER XV

COKE

The characteristics of coke are entirely dependent upon the temperature at which the coal is gasified, as well as upon the method used in its production, and it is also true that the composition of the resultant coke will vary as much as will its physical qualities by the various methods and temperatures employed in its manufacture. Coke is usually divided under two heads, metallurgical coke, the product of the bee-hive, or of the by-product oven, and gas-coke, the latter being a by-product obtained during the carbonization of coal in the gas works.

While the same kind of coal is often carbonized to secure both classes of coke, and while the temperatures employed in both operations often very closely approximate each other, the results often differ to quite some extent, and we therefore find that in the production of metallurgical coke every endeavor is exercised to produce an article which shall possess the requisite strength and purity required in metallurgical work, while the production of gas-coke first implies the manufacture of a gas containing the required calorific and candle powers, the coke being often of secondary consideration.

The tendency prevalent in the manufacture of gas is a raising of the temperatures with the object of obtaining the largest gas yield possible, but, opposed to this, we find the manufacturer of metallurgical coke, using probably the same temperatures, devoting his attention to the production of a coke which will be of sufficient hardness to withstand the weight of the charge in a blast-furnace, as well as the attrition due to the moving of the charge down the column of the furnace. The manufacturer of metallurgical coke is justified in using the higher temperatures, because by their means he is enabled to give the coke its necessary strength, but the same cannot be said of the gas manufacturer, as higher temperatures do not always lead to high candle or calorific power, but rather to a large gas yield, which must often be enriched by some other means.

The late PROF. VIVIAN B. LEWES[1] says:

> "Among the factors that lead to the commercial supremacy of a country by far the most important is the command of fuel or other source of power; and England's position in the past has been governed largely by her coal fields, which in little more than a century raised her to the forefront as a commercial power. The very abundance of our coal supplies was a source of weakness, as it led to outrageous waste, polluted our atmosphere to a criminal extent, and so encouraged uneconomical methods of using it as seriously to deplete our available stock, the result of which has been the increase in price during the last few years, and the certainty that the future will see further advances but no fall to the old rates. The day of cheap coal has gone, never to return."

If this statement is true for England, how much more so is it applicable to our own conditions, where so much remains to be done in conserving our fuel beds, and this conservation can be greatly aided by a concerted effort to educate the user in the benefits to be derived from the burning of coke, but here again we find that in the manufacture of metallurgical coke the warning has not been heeded, and this industry, in many instances, exhibits a prolific waste which is almost criminal. The change from bee-hive to by-product ovens is very slow, but it is progressing; we find that in 1893 the bee-hive ovens produced 9,464,730 short tons of coke, the by-product ovens producing only 12,850 tons during this same period. In 1901, the product was: bee-hive ovens, 20,615,983 tons, by-product ovens, 1,179,900 tons; in 1910, bee-hive ovens, 34,570,076 tons, by-product ovens, 7,138,734 tons; and in 1913 the bee-hive oven product amounted to 33,584,830 tons while that from the by-product ovens was 12,714,700 tons. It is true that the total consumption of coke has increased, but while that produced by the bee-hive ovens in 1893 amounted to 99.99 per cent. of the total, this was decreased to 94.59 per cent. in 1901; to 82.88 per cent. in 1910; and to 72.54 per cent. in 1913.

We also find a great wastage of fuel in the fact that at many plants the so-called "breeze" is not used for any purpose, and immense piles of this fuel can be found in the bee-hive regions, no effort having been made to create a market for it, and this despite the fact that this size coke is admirably adapted for fuel in the hot-air furnaces used in heating residences. In many of the by-product oven plants this breeze is worked up and sold for domestic purposes, and the very fine material, which would be too small to burn in the usual furnance is sold to zinc-retort operators. This lack of foresight in the bee-hive oven operators is probably due to the thought of the consumer that "coal is still so cheap that it is not necessary to find a substitute."

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[1] Jnl. of Gas Ltg., Jan. 26, 1915, page 201.

We find considerable variation in the external appearance of coke, and this also holds good as regards its chemical composition and its physical properties. The color of the coke is due to several causes, such as the nature of the coal used in its production, the temperature maintained during the carbonization process, and the watering, or quenching of the coke; consequently we have coke colors such as dull gray, silvery, or light gray, when coated with a deposit of carbon, this latter color being usually described as of a "metallic luster," and black. The silver gray color is usually secured when the coke is watered in closed vessels, this applying generally to by-product coke, and the latter is thereby made comparable with bee-hive coke, and the coke is usually blackened when it is watered on the wharves of a by-product oven, or in cars, and if it is exposed to air during the watering process. Coke also usually possesses a rough surface, except in those portions of the charge which were in contact with the walls of the retort, these latter portions often presenting a smooth, glassy surface, while the hair-like threads which often appear on the surface are also due to a deposit of carbon coming from the heavy hydrocarbons, and which is caused to be given off when the coal is coked at low temperatures.

The physical structure of coke also varies in many particulars, as it may be light and porous, or it may be heavy and dense; again it may be soft, or it may be hard and of such structure that it will be capable of sustaining the heavy load produced during metallurgical operations; it may also be quite brittle, and will then break into small pieces while being handled. Some grades of coke give forth a clear ringing sound when sharply struck, while others give a heavy, dull sound, the latter being due to a soft structure. If the cell space is small, the coke will be dense, but the size of the cells will not govern the porosity and the terms porosity and density refer only to the cell space, while the terms hardness and softness apply only to the walls of the cells in the coke.

C. A. MEISSNER states that good physical coke tests for a standard quality of blast-furnace coke is exhibited by the following:

Mixture	Shatter test		Specific gravity		Porosity or percentage of cell space
	Through 2-in. screen	On 2-in. screen	Apparent	True	
80 per cent. Pocahontas 20 per cent. Ronco	15.6	84.84	0.976	1.841	47.03
80 per cent. Pocahontas 20 per cent. Ronco 1, Ills.	16.11	83.89	0.950	1.824	47.92
60 per cent. Pocahontas 40 per cent. Ronco.	14.06	85.94	0.992	1.834	46.31

In all by-product ovens the coke is pushed by means of a ram, of very heavy cross-section, usually so arranged as to carry a rack of heavy cast-steel teeth on its bottom; this rack engages a pinion, and is thereby driven into the oven, power being supplied by a motor of from 40 to 50 hp., although some rams are driven by steam, and others are operated hydraulicly. The ram is usually guided by a series of rollers, so spaced as to maintain the ram in a central position, and the entire machinery is supported on a heavy steel frame running on two tracks, the same motor used for operating the ram being employed to traverse the machine in front of the ovens. The operator's cab, containing all of the controller levers, is so situated that the operator has a clear view of the oven and its immediate surroundings. This machine also usually carries a drum and cable for hoisting and lowering the oven doors.

Coke Quenching and Loading.—The treatment of by-product coke often presents some very difficult operations, these difficulties often being increased by the short time elapsing between the pushing of the various ovens. At Joliet, Ill., the quenching bench is located immediately in front of the ovens, the coke being sprayed with a definite amount of water as it emerges from the oven, and, after steaming, the mass of coke is watched closely; if any red-hot coke should appear, it is quickly quenched by means of water from a small hand hose. This method of quenching permits of three or four charges lying in front of a battery, and also allows sufficient time to thoroughly quench the coke without soaking it with water, thus ensuring a reasonably low and uniform moisture content. This system of quenching, however, possesses the objection of forming a heavy mass of steam directly in front of the oven.

In many instances quenching cars are used, the coke being pushed into the cars, the latter being then transferred by a motor or locomotive to a permanent quenching station, where the coke is properly sprayed with water, as it is very objectionable to quench the coke in cars directly in front of the ovens. At those plants where the coke is transferred to a quenching station, the coke is permitted to remain in the cars until the steam has been given off, after which any remaining red hot coke is quenched by a hand-hose. Again we find that in some instances the coke is partially quenched in the cars, and then transferred to inclined platforms or pockets; at these pockets hand-quenching is resorted to, after which the quenched coke passes on to the screening plant.

At Farrell, Pa., the hot coke is pushed into inclined pockets, these pockets having sufficient capacity to hold a number of charges, this capacity also permitting of sufficient time to allow of proper quenching and steaming-off; the coke passes from the pockets onto a belt conveyor, which carries it on to the screens.

The system adopted at Duluth embraces a quenching platform located away from the ovens, and the coke, which has received preliminary treatment in the cars, is transferred to this platform, where the process is finished.

The methods of treating coke from by-product ovens in Europe are somewhat different from those practised here, and the machines used, perhaps somewhat modified to suit our conditions, might be adopted by American operators with success, and reference is here made to the descriptive articles by THAU on coke handling and quenching equipment.[1]

The simplest system of loading is that of Coulson and Company, of Spennymoor, which consists of a belt conveyor operated on a system of three tracks, the belt extending from the coke wharf to the middle of the railroad cars. The belt, driven by means of an electric motor, is run in front of the quenched coke, the latter being shoveled onto the belt by hand, using coke forks. The objection to this method of loading is due to the interference of the pile of coke with the traverse of the belt, thus preventing the loading of the coke into the cars in sequence as the ovens are pushed. The BAGLIN machine, Fig. 104, is intended to overcome this objection; the coke belt reaches some distance over the wharf in one direction, and extends to the cars in the other, but the belt is located somewhat higher above the wharf than in the COULSON arrangement, and this additional height gives the machine greater freedom of movement. Both of these arrangements require quite a number of laborers, and do not reduce operating costs to any extent; they can readily be replaced by a wharf which will permit of dropping the quenched coke through openings into cars on tracks located beneath the wharf.

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[1] Glückauf, Sept. 2, 1911, pages 1361–71. Sept. 9, pages 1408–14. Sept. 16, pages 1440–45. Feb. 28, 1914, pages 321–30. March 7, pages 365–76, and Stahl und Eisen, March 19, 1914, pages 495–98. Jan. 7, 1915, pages 18–20. Jan. 14, pages 48–49.

The loader built by GRÉGOIRE, of Seraing, is of quite a different construction; the coke wharf is provided with two sets of tracks

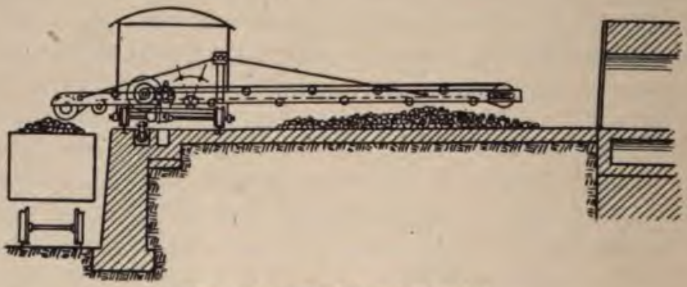

FIG. 104.—Baglin coke loader.

so arranged that the loading machine, which has a plow-like action, passes along the outer set of tracks first, pushing the coke over chutes into the waiting cars, Fig. 105, after which it returns and is switched onto the tracks lying directly in front of the battery. The machine now moves this portion of the coke forward, so that a final trip on the outermost tracks transfers the entire coke pile into the cars. The machine is so arranged that the plow-like pusher can be elevated, thus allowing the small coke to remain on the platform until the large coke has been pushed away.

Some of the mechanical quenching devices used in Europe, require that the coke wharf have an inclination of at least 1:10, in order that the large amount of water used may not come into contact with the ovens, thus preventing injury to the latter. The simplest mechanical quenching device, used in Europe, consists of a series of perforated pipes, suspended from an overhead trolley, and which is run in front of the oven which is being pushed; the coke, as it emerges from the ovens, is thus pushed between these two sets of pipe, and thoroughly quenched by many fine streams of water, but great care must be exercised to see that the small nozzles in the pipes are not clogged up. With a device of this kind the coke cannot be pushed at a speed greater than 6.5 ft. (2 meters) per minute if thorough quenching is desired. In order to prevent the access of air to the coke, the DARBY quencher, Fig. 106, is supplied with steel plate walls, located between the perforations in the pipes, this addition greatly assisting the quenching process. Another system of this character proposes to use a sort of tunnel, which becomes a continuation of the oven proper, the tunnel-like structure being provided with double walls, the space between these walls being filled with water which is projected onto the hot coke through numerous openings; the weight of this latter device requires that it be moved in front of the battery on

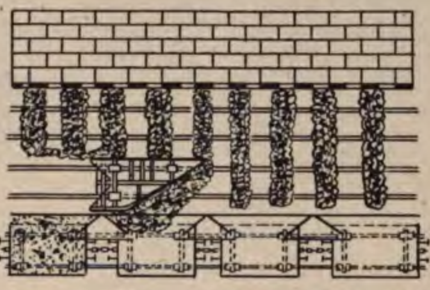

FIG. 105.—Plan of Grégoire coke loader.

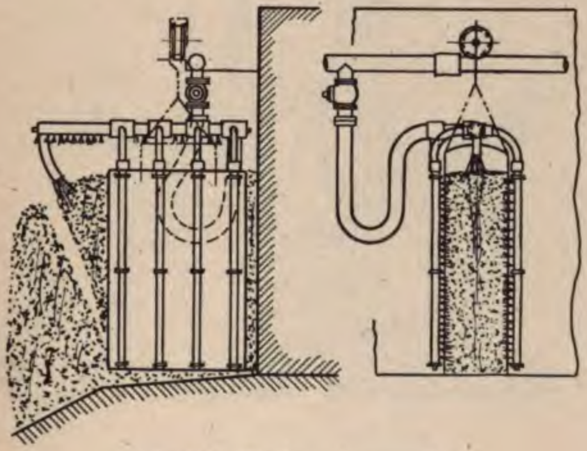

FIG. 106.—Darby coke quencher.

depressed floor rails, instead of being suspended from a trolley as explained above. All of these devices are very lavish in the expenditure of water.

A greater inclination to the coke wharf, say from 15 to 30 degrees, as shown in Fig. 107, greatly assists the loading process because the pushers need only to move the coke over the short horizontal space in front of the battery, the coke sliding down the incline and into the cars by gravity. Either mechanical or hand quenching can be applied to this case, and the coke can be loaded in the cars by means of coke forks, or the ramp can be provided with chutes to direct the flow of coke, but the entire arrangement requires that either the ovens be built on a considerable elevation, or that the railroad tracks be placed at a rather low level.

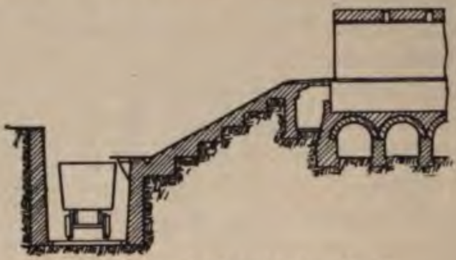

FIG. 107.—Inclined coke wharf.

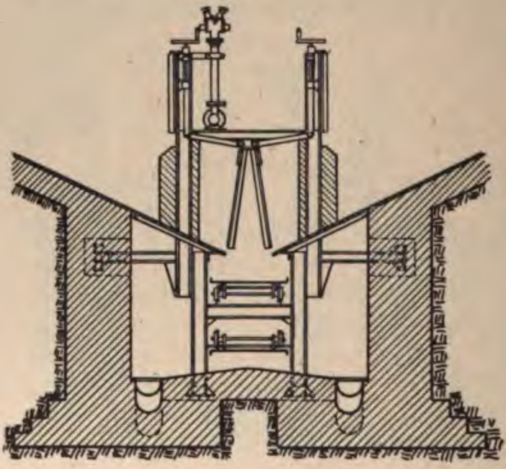

FIG. 108.—Loading valves on ramp.

The ovens at one plant in England are located at such an elevation that it is possible to have the lower side of the inclined ramp deliver the coke directly into bunkers, the water being withdrawn by means of gratings at the lower end, the water passing through same and into water troughs. Undue breakage of the coke, produced by tumbling down the ramp, is hard to avoid, but several partially successful attempts have been made in this direction.

In several of the later European installations, the coke passes over a 30- to 40-degree ramp and delivers itself onto a conveyor, the end of the latter being so inclined as to deliver the coke directly into the screening plant. The lower end of the ramp is provided with adjustable loading valves, and these are not opened until the coke has been thoroughly cooled. These loading valves are located along the entire length of the battery, as shown in Fig. 108, or they may be attached to a movable frame, running on tracks, which can be moved to a point in front of the oven which is being pushed. The disadvantage possessed by this arrangement consists in the possible failure of the belt conveyor, which at times receives hot coke, and in such an event the entire process may be temporarily disabled.

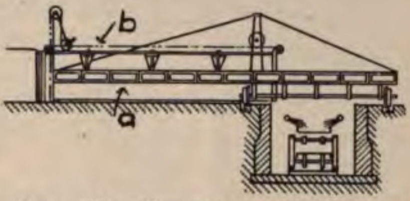

Fig. 109.—Burnette coke quencher and loader.

The Burnett quencher and loader is shown in Fig. 109, this machine being applied to horizontal coke wharves; the lower portion of this device (*a*) consists of a casing made of heavy buckle plates, the perforated water spray pipes being located and protected within the buckle. After the coke has been pushed from the oven into this casing, a piston is lowered on the oven side and, after the coke is quenched, the casing is pushed forward on the rollers (*b*), the contents being thus slowly passed over the wharf and brought to the loading point.

The Méguin loader is so designed that it can be moved along the battery, Fig. 110, the discharged coke passing down the ramp and onto the conveyor (*a*); this conveyor elevates the coke and discharges it over the grizzly screen (*b*) and chute (*c*) to the cars, the resultant breeze being deposited in the bin (*d*), from whence it is drawn into the larry cars (*e*).

A double screening of the coke is accomplished by the Allport loader, Fig. 111, where the coke previously quenched in (*a*) is first screened over the incline (*b*), then deposited by the chute (*d*) onto the conveyor (*c*), the latter being of woven wire, thus giving the coke its final screening and delivering the large coke to the cars. Due to this device the coke wharf can be constructed of very narrow dimensions, but, as a final quenching is not included, burning coke, with its consequent disagreeable results, is often deposited in the cars.

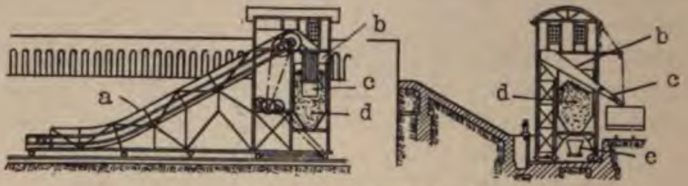

Fig. 110.—Méguin coke loader.

If the plant is to be constructed without any coke wharves at all it becomes necessary to so construct the quenching and loading devices that they cannot be damaged by either heat or steam from the quenching process. The device shown in Fig. 112, and which is self-explanatory, was built by the Wellman-Seaver Company some years ago for the Lackawanna Steel Company at Buffalo, but as this device does not include an arrangement for screening the coke, its application is very limited. The Moore device gives the coke better treatment than the Wellman-Seaver

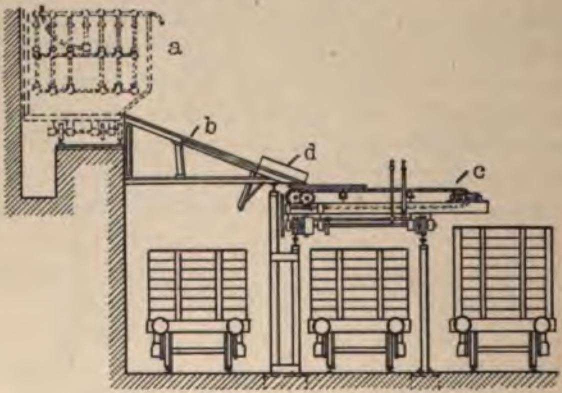

Fig. 111.—Allport coke loader.

and results in producing a silver-gray coke similar to that produced in the bee-hive oven. This device consists of a closed chamber, similar to that shown in Fig. 109, which is movable in front of the battery; this chamber is just as long as the oven and of a cross-section which permits of taking in an entire charge, the coke being quenched within the chamber, the resultant vapors escaping through a short stack. After the coke has been quenched, the machine is run to a discharging point, at the end of the battery, where the coke may be delivered to cars or to a screening plant.

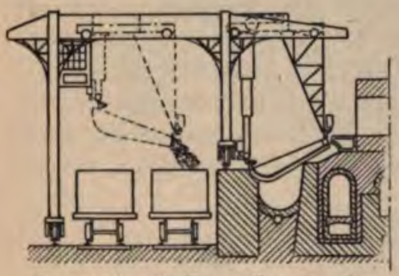

Fig. 112.—Wellman-Seaver coke loader.

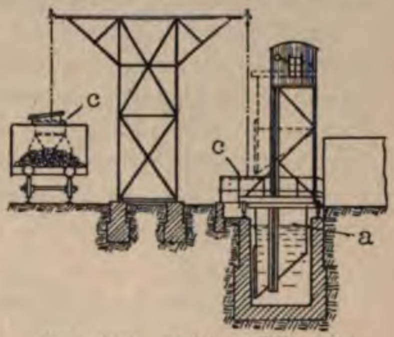

Fig. 113.—Bleichert-Illig coke quencher and loader.

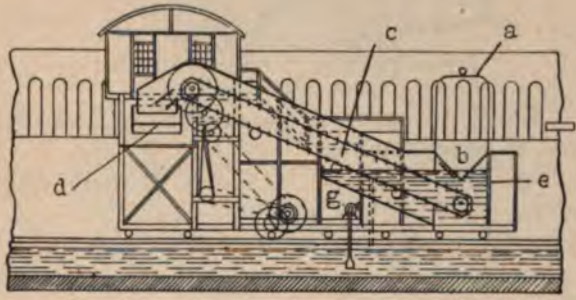

Fig. 114.—Méguin coke quencher and loader.

The **Bleichert-Illig** machine. Fig. 113, makes use of a water trough, located in front of the ovens, and into which the perforated containers (*a*) filled with red hot coke, are dipped, these containers being movable along the front of the battery. After the container has been sunk into the water and the coke thus quenched, it is raised to a position which permits of depositing the coke into the buckets (*c*), the latter being then raised on the crane-like structure, transported to the other side, and the coke is deposited in the cars. A modification of this machine consists in using the perforated container as a loader, and as but little water is used, dry coke is produced, this latter condition being assisted by the temperature of the water.

The MÉGUIN device, Fig. 114, belongs to that class of apparatus which does not require a coke wharf, and in which quenching, screening, and loading is a continuous mechanical operation. The coke is pushed from the quencher (*a*) into the hopper (*b*), from whence it is delivered, under water, to the conveyor (*c*), the latter carrying the coke to the screen (*d*), from whence it is loaded. The overflow water from the container (*e*) flows into a trough located in front of the battery, and is pumped from there back into the container as required.

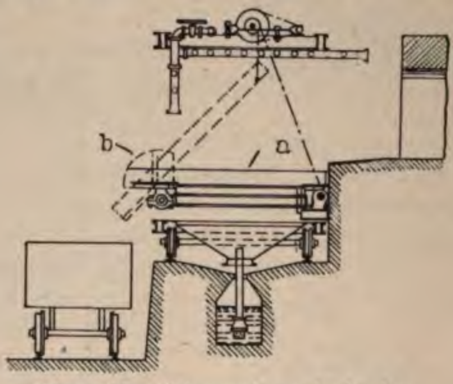

FIG. 115.—Schumacher coke loader.

The SCHUMACHER device, Fig. 115, is of quite a different construction; the coke is pushed from the oven on to the container (*a*), the gate (*b*) being closed during this period. The bottom of the container is made of reciprocating screen bars, thus effectively screening the coke before the rear end is elevated, the gate (*b*) opened, and the coke deposited in the cars. The coke is quenched in the container, the water passing into the trough located below the machine. The MEYER device is a modification of the SCHUMACHER, and it consists of a perforated, shovel-like container, into which the coke is pushed, the quenching sprays reaching the coke from both top and bottom, excess water being drained off through gratings at the front end; in order to load the coke into cars, or to deposit it on the coke-pile, the rear end of the container is raised, and the coke is spilled out of the forward end.

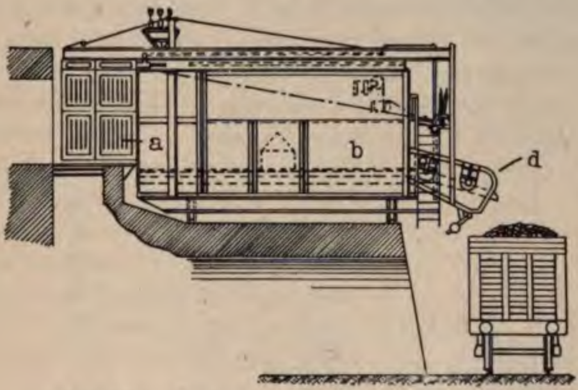

FIG. 116.—Goodall coke quencher and loader.

The GOODALL device, Figs. 116 and 117, receives its coke from the oven through the quencher (*a*), the coke being thus pushed out onto the revolving table in the casing (*b*), the door (*c*) being closed during this period. After the coke has been quenched and

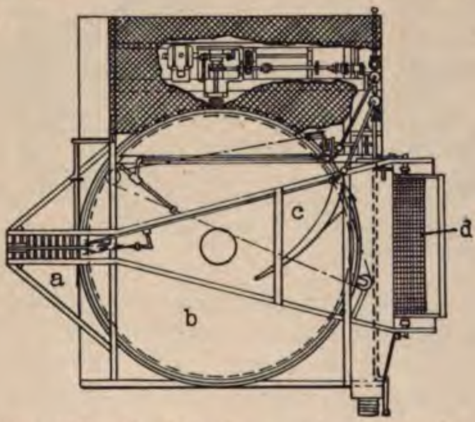

FIG. 117.—Goodall coke quencher and loader.

cooled, the plow-like door (*c*) is opened, the table is revolved, and the coke is pushed out onto the shaking screen (*d*), and from thence to the cars. It is stated that this machine will quench and load a complete-oven charge in 27 minutes; its construction is simple, and its weight only 28 tons. The revolving table is covered with cast-iron grid plates. One of these machines operating at Spennymoore, where it serves a battery of 60 ovens, producing 1700 tons of coke weekly, requires from two to three men as against the 21 to 27 men formerly required for hand quenching and loading, the waste amounting to 0.5 per cent. and the moisture in the coke to only 2 per cent. This machine was modified by Méguin in such manner that it no longer travels on a ramp, but on a depressed track, as shown in Fig. 118, the entire machine being contained in and supported on the frame (*a*). The original plow-like door has been replaced by a scraper (*c*) operating on the table (*b*), the scraper being held in place by the pawl (*d*) while the table is being turned; the coke is thus pushed out over the grizzly (*e*) and chute (*f*) into the cars.

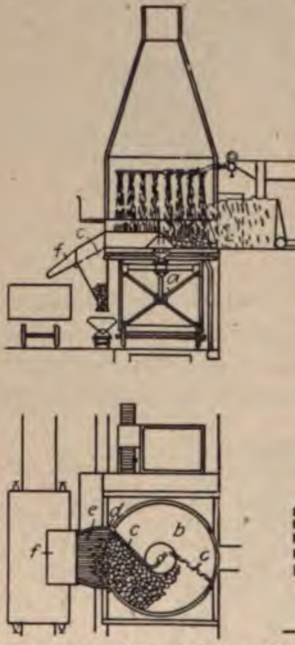

Fig. 118.—Méguin's modification of Goodall's machine.

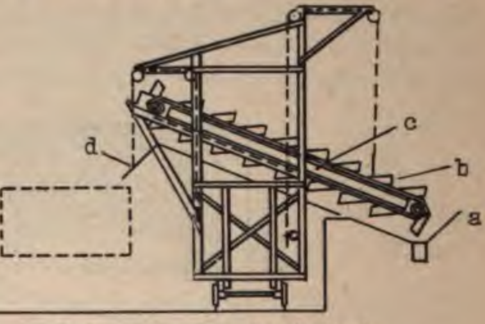

Fig. 119.—Brunck coke loader.

The Brunck loader, Fig. 119, takes the coke delivered by the inclined ramp (*a*), up with the buckets (*b*) of the conveyor (*c*) and delivers it over the chute (*d*) to the cars. Mechanical quenching not having met all that was expected of it, this method has been almost universally succeeded by hand quenching, and for this purpose the loader is usually moved slowly along the battery while the coke is being pushed and quenched, while in other installations the coke car is moved.

The mechanical loading of coke is greatly simplified if the ovens are built on such an elevation as will permit of placing the discharge end of the ramp some distance above the top of the railroad car. The screening and loading machine of HUMBOLDT, Fig. 120, is well adapted for such operation. When the chutes (*a*) are opened, the coke slides onto the reciprocating screen (*b*), which screens and carries it to the car (*c*), the breeze falling into the hopper (*d*) from whence it is drawn from time to time into larry cars located on tracks beneath the hopper bottom.

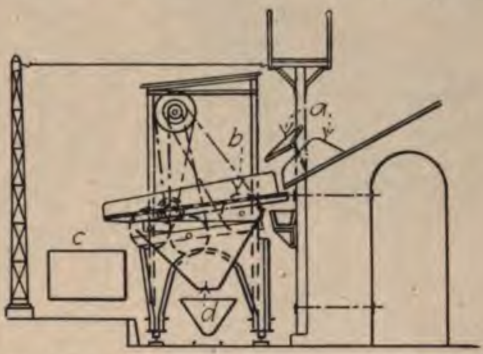

FIG. 120.—Humboldt coke screener and loader.

Another machine, built by MÉGUIN, is shown in Fig. 121; here the coke, after the chute (*a*) is opened, passes over the flat screen (*b*) into the cars (*c*), while in the GRÖPPEL machine, Fig. 122, the coke is screened and transferred to the cars by means of an eccentric screen, the individual bars of which are set in motion by eccentrically operated drives.

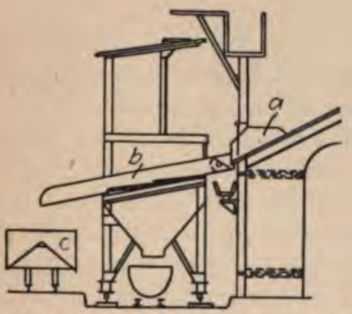

FIG. 121.—Méguin coke screener and loader.

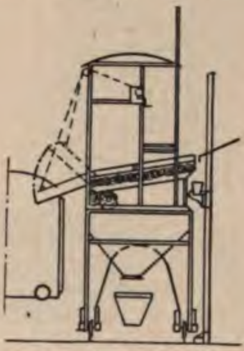

FIG. 122.—Gröppel coke screener and loader.

A machine of somewhat different type is required for those ovens which are elevated but slightly above the ground, such machines being designed to raise the coke, screen it, and load it into cars, and they are usually of very large and heavy construction. The Fabry machine, Fig. 123, is an example of this class. The horizontal portion of the scraper conveyor (*a*) receives the quenched coke from the ramp and carries it up the incline (*b*), a breeze hopper (*d*) being located below the incline, the breeze being passed through the grate-like bottom of the inclined conveyor. The scraper conveyor is returned over the incline at (*c*) after it has deposited the coke in the loading chute (*e*).

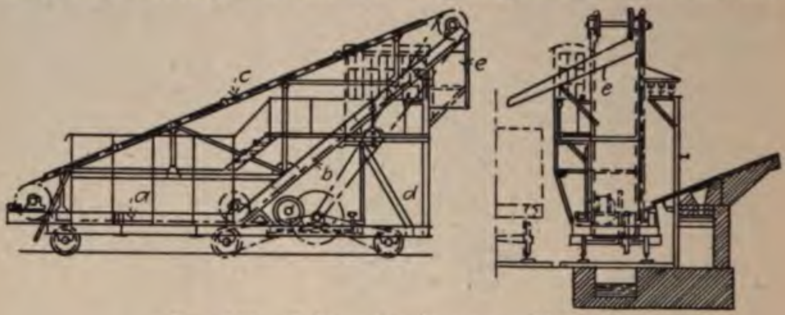

Fig. 123.—Fabry coke screener and loader.

Another machine, designed by Coppée, which is applied to such ovens whose ramp lies below the ground surface, is shown in Fig. 124; here, after the loading valves have been opened, the coke from the ramp (*a*) is deposited in the bucket (*b*), which is then raised and its contents dumped onto the bars of the shaking screen (*c*), the breeze passing into the hopper (*d*), while the large coke is transferred to the tilting trough (*e*), the latter serving two lines of cars.

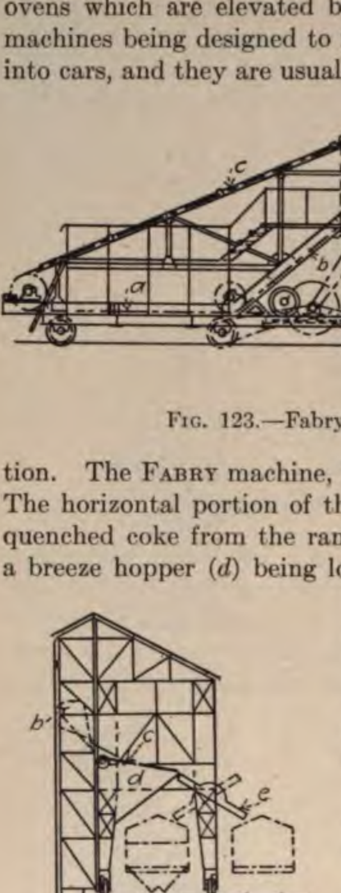

Fig. 124.—Coppée coke screener and loader.

Fig. 125 shows a quenching machine, built by Koppers, in use in connection with the Koppers chamber ovens at the Leopoldau gas works, Vienna. Here the coke is pushed out of the oven (*a*) into the quenching chamber (*b*), and from thence, after opening the gates in the bottom of this chamber, over the incline

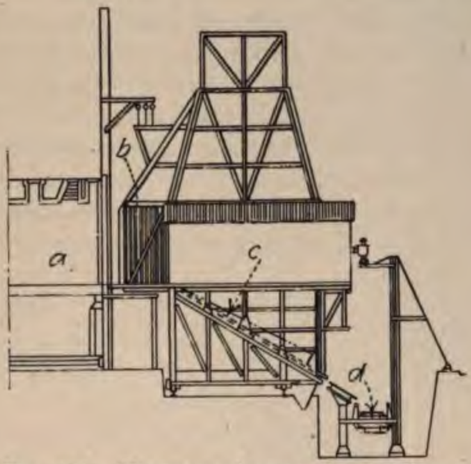

FIG. 125.—Koppers coke quencher at Vienna.

(*c*) onto the conveyor (*d*), the latter transporting the coke to the screening plant.

The TIGLER quencher and loader is shown in Fig. 126; the coke

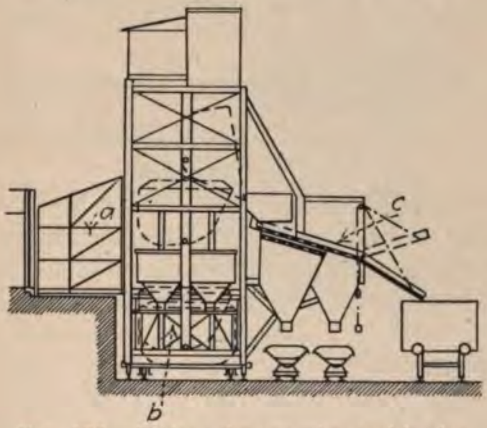

FIG. 126.—Tigler coke quencher and loader.

coming from the oven passes through the deflecting shield (*a*), and from thence into the lower bucket (*b*), the latter being built with double walls, the inner bottom of which is perforated, thus permitting the water between the walls to rise up through the holes in the inner bottom and flood the hot coke. After the coke has been quenched, the bucket is raised and its contents are dumped onto the screen (*c*), passing from thence to the cars.

In the **Bamag** quencher, screener and loader, Fig. 127, the coke is directed by the frame (*a*) into the bucket (*b*), the latter dipping its greater portion into the water-tank (*c*), thus cooling the coke. After the entire charge has been passed into the bucket, the latter is lowered, and a gate in its bottom is opened, thus permitting the water to enter the bottom through a grid plate. After a short period of immersion in the water, the bucket is raised, the water is drained off, the bucket is tilted, and the coke is dumped into the hopper (*e*). This hopper is provided with an automatic feeder which distributes the coke onto two grid conveyors (*f*), the latter having a wave-like motion, being thus screened and, at the same time, carried to the cars.

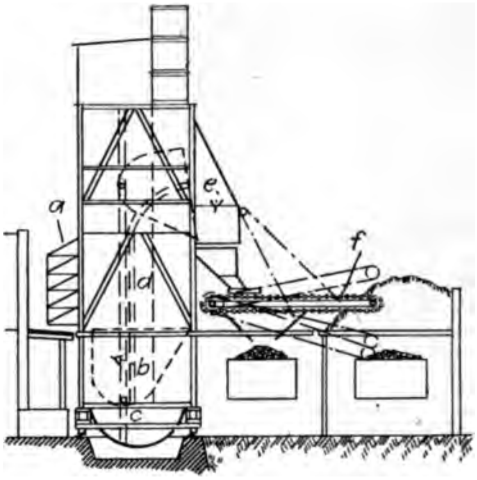

Fig. 127.—Bamag coke quencher, screener and loader.

A machine constructed in a similar manner, but in which the quenching bucket is stationary, is built by **Méguin**, while the one built by **Koppers**, for the Budapest gas works, does away with a quenching water reservoir entirely. The water is pumped by the pump at (*a*), Fig. 128, from the trough (*b*) into the double-bottom of the bucket (*c*), rising from thence through the coke mass. The vapor stack (*d*) with the deflecting frame (*e*) operates on a separate set of tracks; the quenching bucket (*c*) is run to a station where it is raised and dumped.

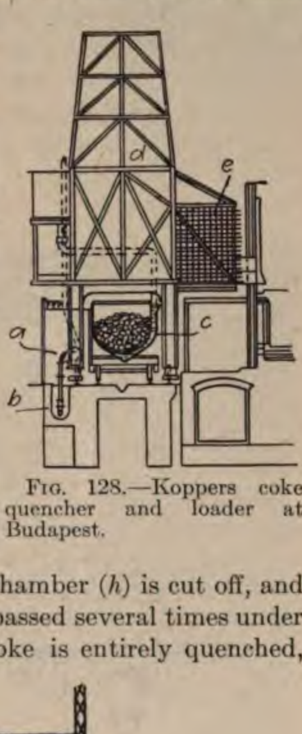

Fig. 128.—Koppers coke quencher and loader at Budapest.

The quenching and loading installation at See-brügge is shown in Fig. 129, and it consists of two separate moving units, the coke car (*K*) and the quenching car (*L*). The quenching car is connected at (*s*) to water plugs distributed at regular intervals in front of the batteries, and the coke is quenched by means of the shower at (*b*), as well as in the chamber (*h*), the latter being constructed similar to that of Darby. During the period of coke pushing, the coke car, about 27 ft. long, is slowly moved along in front of the oven, so that the coke will be deposited over its entire length. The inclined bottom (*w*) of the coke car is covered with a grating made of steel bars, which latter draws off the breeze. After the coke has been pushed, the water in the quenching chamber (*h*) is cut off, and the car with its content of hot coke is passed several times under the shower sprays (*b*) or until the coke is entirely quenched, after which the car is run to one of the two loading stations, the gates (*k*) are opened, and the coke is dumped into waiting cars.

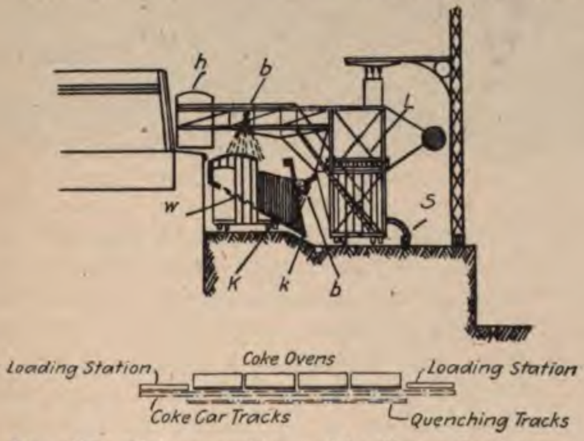

Fig. 129.—Coke quenching and loading at See-brügge.

Another quenching device used in Europe consists of a tower erected over a concrete water tank, the latter being sunk in the ground. The coke buggy, which is of the open lattice type, the body being made separate from the frame, is filled with coke and run under the tower, where the coke-filled body is lifted from the frame and plunged into the water, remaining submerged for about 8 seconds; it is then raised, replaced on the wheel frame, and run into the coke yard. The entire mechanism for raising the bodies of the buggies is hand operated, and it consists of an endless chain which is caused to pass over a wheel fixed to the end of a spindle, the latter being provided with pinions for carrying the chains, one end of each chain being supplied with lifting hooks, while the other end is provided with a counterweight. It is claimed that coke quenched in this manner is not saturated with water, and that if the lumps are broken immediately after quenching, their interiors will be found quite hot, but that incandescence and combustion of the outer layer are completely stopped.

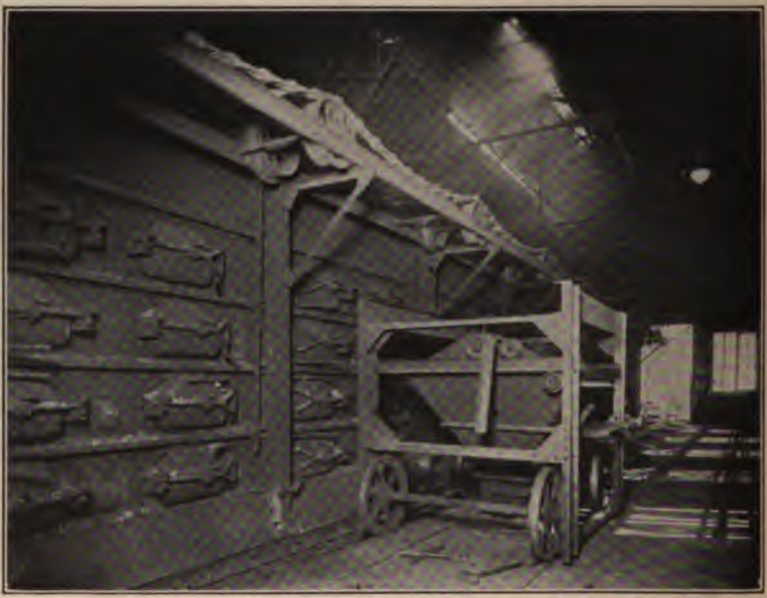

FIG. 130.—De Brouwer conveyor at St. Louis, Mo.

The ILLIG system of quenching is somewhat similar to the one described above; a perforated, steel box is filled with coke as it is discharged from the retort, being then sunk into a water tank; after the coke is quenched, the box is raised, the water draining off as it rises, and then run off to the coke yard, the coke carriers being arranged as telphers on overhead rails.

C. E. ARNOLD devised a method for quenching coke by which a high moisture content is to be avoided. In this system the coke is discharged into a closed vessel, the latter having a perforated bottom; air is forced up through these perforations, and in its heated state is returned to a lower chamber, provided with water coils, where it is cooled and then passed back through the hot coke. The coke chamber is provided with tight-fitting doors.

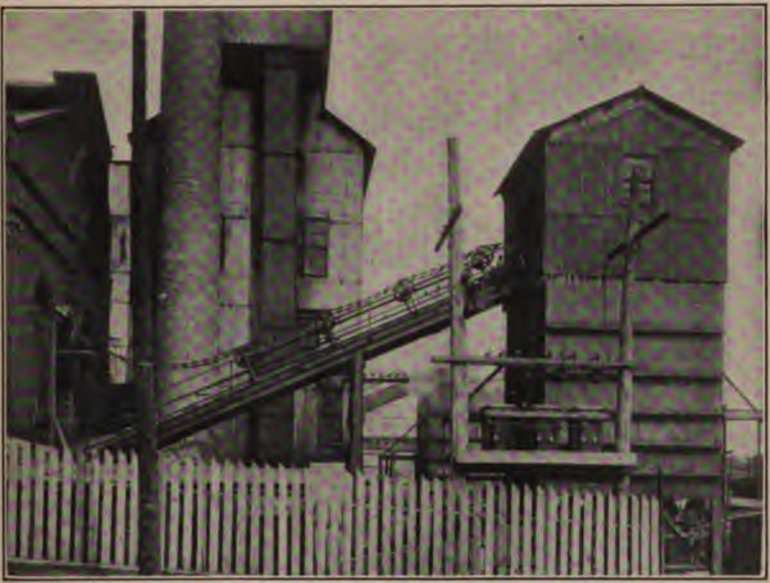

FIG. 131.—Inclined De Brouwer conveyor at St. Louis, Mo.

In gas works practice in Europe the coke is usually quenched in a DE BROUWER hot coke conveyor, and this method of treating coke is finding general favor here. The DE BROUWER conveyor consists of a trough, made of steel or cast iron, containing a moving double-strand chain, the two chains being connected by cross-bars at regular intervals, these bars acting to carry the coke forward. Quite a number of different style troughs and chains are being used in America, but whether the trough is constructed of steel or cast iron, it should be protected with hard white-iron removable bottom plates and sides. This conveyor is driven by means of specially designed sprocket-wheels, directly connected to a motor, or by means of a driving chain between the motor and the gears. The conveyor is usually provided with quenching sprays at each bench, a final spray being located at the point where the conveyor leaves the house and rises to the point of deposit. These conveyors are also usually provided with water-return troughs, which form lateral enlargements of the trough, by means of which the water carried along with the coke can be returned behind the latter, thus ensuring a quenching of the coke without subjecting it to a drenching action. The elevating portion of the conveyor should be so arranged that all surplus water is positively drained from the trough, this arrangement ensuring a coke delivery which shall be as free from moisture as possible. The De Brouwer conveyor is used in connection with horizontal, inclined, or vertical retorts, and the chain can be returned either overhead or below the conveyor, depending upon the arrangement of the producers.

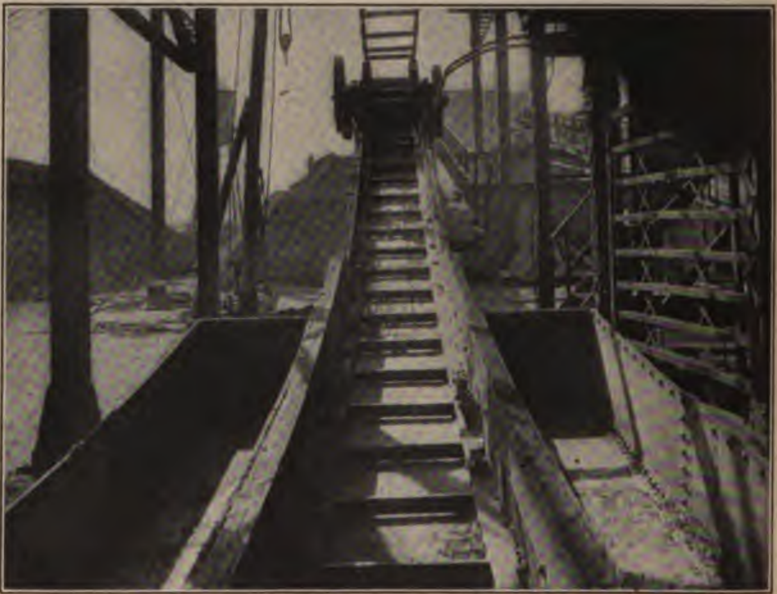

Fig. 132.—Delivery end of De Brouwer conveyor at Providence, R. I.

Fig. 130 shows the De Brouwer chain in the retort house of the Laclede Gas Light Company at St. Louis, while Fig. 131 shows the chain as it emerges from the house and delivers the coke to the storage bins. Fig. 132 shows the De Brouwer chain at Providence as it ascends the incline, and Fig. 133 shows the same chain out in the yard at a point where it delivers the coke to the concrete storage bins.

It is seen from the above that quite a number of quenchers, loaders, and screeners of various designs are in use in Europe, but it is questionable whether any one of them give complete satisfaction, as the proper preparation of coke requires more of the "human element" than is possible with these mechanical contrivances.

The economical treatment of coke requires that it shall be handled in a manner which will reduce the breakage to a minimum, and care must be taken to see that no burning pieces are left in the mass, while, at the same time, the quenched coke should not contain more than 2 or 3 per cent. of moisture. An examination of operating conditions existing with mechanical quenching devices, shows that, as a rule, the machines push the coke from the oven into a very narrow space, thus permitting the top of the charge to cool while the bottom remains red hot; if sufficient water is supplied to quench this bottom layer, we find the top layers drenched, and considerable air drying will be required to evaporate this surplus water. Again, those machines which make use of screens or perforated bottoms, often permit of such an access of air to the coke that it will again take fire, unless overquenching has been practised.

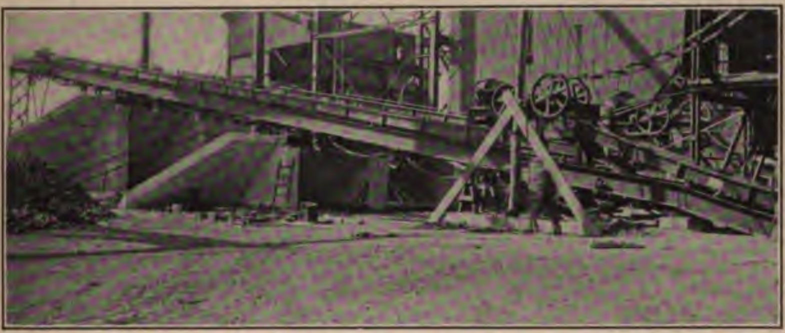

Fig. 133.—De Brouwer conveyor at Providence, R. I.

The cost of maintaining these machines is very heavy and it is questionable whether they are really economical as compared with hand quenching.

The final condition of all coke is probably due, primarily, to the method of quenching, and this permits of making a dry, silvery coke, or a dull black coke, and while the silvery appearance does not add any intrinsic value to the coke, it does assist in removing old-time prejudices and helps the sale. We find that combustion begins as soon as the hot coke is pushed from the oven, and unless it is immediately quenched in a proper manner, the appearance will be greatly affected; only sufficient water should be applied to the coke to arrest combustion, as a greater quantity will only drench the coke with no good result.

Coke Screening.—The handling and sizing of the coke is the next important step, and here again we find many diverse opinions. After the coke has been quenched it is usually raised to the screens, belt conveyors, or large hoists finding much favor for this service. Elevating the coke by means of a belt ensures a more even distribution on the screen than will be the case if the entire content of a large bucket is dumped at one time. Belt conveyors have been the subject of much criticism on account of wear, but they can be depended upon as far as a minimum coke breakage is concerned. MR. J. W. SHAEFFER[1] inclines strongly to belt conveyors, because of the delicacy with which coke can be handled upon them, and because of their reliability due to the absence of intricate mechanical devices. Coke should be delivered to a belt in such manner that coke will slide on coke, thus avoiding a precipitate deposit of the coke on the belt with consequent abrasion. In this relation SHAEFFER prepared a table showing the angles of repose and slide of coke upon coke, this table being of considerable value to all coke operators.

An examination of this table shows plainly how both the angles of repose and the angles of slide decrease with the size of the coke.

Belt conveyors are best fed by means of either rotary or reciprocating feeders, thus ensuring the conveyor against an overload, and belts usually operate best under a constant load condition. This also holds good for crushers and screens, where an avoidance of fluctuating feed is greatly to be desired.

The proper feeding of a belt conveyor is absolutely necessary for its proper operation, and the object to be achieved in designing any feeding arrangement is to see that the coke is deposited on the belt in the same direction at which the belt is moving, and, if possible, at the same approximate speed as that of the belt's travel. The feeding arrangement should be so designed that the bulk of the coke shall be delivered to the center of the

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[1] Proc. Am. Gas Inst., 1913, page 415.

TABLE LXXXVII.—ANGLES OF REPOSE AND SLIDE IN DEGREES OF COKE UPON COKE. SHAEFFER

Foundry:	Test	1	2	3	4	5	6	7	8	9	10	Average	Moisture
Regular	Repose	44	43	44	44	43	44	44	43	44	44	43.7	0.98
	Slide	47	46	46	47	46	47	46	46	46	46	46.2	
Dry	Repose	44	43	43	44	44	44	44	44	44	44	43.8	0.00
	Slide	47	46	46	46	46	45	45	46	46	46	46.0	
Wet	Repose	43	43	43	44	44	44	44	44	44	44	43.7	
	Slide	46	46	47	46	46	46	46	46	46	46	46.1	8.00
Blast-furnace:													
Regular	Repose	45	44	45	45	44	45	45	43	44	45	44.5	1.40
	Slide	48	47	46	47	47	47	47	47	46	47	46.9	
Dry	Repose	45	46	44	43	44	44	45	44	44	44	44.3	0.00
	Slide	47	48	46	47	46	47	47	47	47	47	46.9	
Wet	Repose	42	43	43	42	44	43	43	43	43	44	43.0	8.00 to 10.00
	Slide	45	46	45	47	46	46	46	46	46	46	45.8	
Egg:													
Regular	Repose	39	40	39	40	40	39	40	40	40	40	39.7	3.10
	Slide	42	43	42	42	42	42	42	42	42	42	42.1	
Dry	Repose	39	41	40	40	40	39	40	40	40	40	39.9	0.00
	Slide	42	42	43	42	42	43	42	42	42	42	42.2	
Wet	Repose	41	40	39	40	41	41	40	41	41	41	40.5	8.00 to 10.00
	Slide	43	42	43	43	42	43	42	43	43	43	42.7	
Range:													
Regular	Repose	41	39	40	41	39	39	41	40	39	39	39.8	6.10
	Slide	44	43	44	42	43	44	44	44	44	44	43.6	
Dry	Repose	41	39	39	40	41	40	39	39	39	39	39.6	0.00
	Slide	44	43	44	43	44	44	44	43	44	44	43.6	
Wet	Repose	41	40	41	39	40	41	41	41	41	41	40.6	8.00 to 10.00
	Slide	43	44	43	44	44	43	43	43	43	43	43.4	
Chestnut:													
Regular	Repose	40	41	40	40	41	40	39	40	40	40	40.1	8.30
	Slide	41	42	42	41	42	42	42	42	42	42	41.8	
Dry	Repose	40	39	39	40	40	40	39	40	40	40	39.7	0.00
	Slide	42	41	42	41	42	42	42	42	42	42	41.8	
Wet	Repose	40	39	40	40	40	40	40	39	40	40	39.8	9.00 to 11.00
	Slide	41	42	41	41	42	41	42	41	41	41	41.3	
Pea:													
Regular	Repose	39	39	38	39	38	40	39	39	39	39	39.0	9.20
	Slide	42	41	41	42	41	41	41	40	41	41	41.1	
Dry	Repose	39	40	39	38	39	39	39	39	39	39	39.0	0.00
	Slide	42	42	41	41	41	42	41	41	41	41	41.3	
Wet	Repose	38	39	40	39	40	40	40	40	40	40	39.6	10.00 to 12.00
	Slide	42	42	41	42	41	42	42	42	42	42	41.8	
Breeze:													
Regular	Repose	36	37	38	37	36	37	37	37	37	36	36.8	12.80
	Slide	42	42	41	42	42	42	41	42	42	42	41.8	
Dry	Repose	37	38	36	38	37	37	37	37	36	38	37.1	0.00
	Slide	41	42	42	41	42	41	41	42	42	42	41.6	
Wet	Repose	39	40	39	41	40	40	40	40	40	40	39.9	12.00 to 15.00
	Slide	43	44	43	43	44	43	43	43	43	44	43.3	

belt, and the belt should be able to free itself and thus permit the load to be properly distributed while coming to a state of rest.

Both fluted roll and tooth crushers give good results on coke, but either type should be so arranged as to permit of a rapid adjustment of the space between the rolls, and this adjustment should remain fixed when once made. The crusher rolls should be provided on one side with springs, thus forming a protection to the crusher in case some hard substance, such as iron, should be passed through; the space between the rolls of crushers provided with safety springs must be somewhat less than when these springs are not provided, as otherwise, due to the flexibility of the springs, the rolls might separate and thus produce an uneven crushing of the coke, but SHAEFFER states that the action of a crusher with these safety springs, or "backing up" springs, under light loads, is to crush a great many pieces of coke already undersize, and under heavy, or overloads, the spreading of the rolls permits much coke to pass through, larger than the maximum size desired. Care should be exercised to see that the coke stream is delivered uniformly over the entire length of the rolls, otherwise the rolls will soon be greatly reduced in size at the point of feeding, and if this condition is not immediately rectified the space will become so great that proper crushing will no longer be possible.

Screening the coke is a process which is often attended with much difficulty, as proper screening does not only mean that the coke shall be separated into its various sizes, but that the breeze which clings to the various pieces shall also be removed to the greatest possible degree. SHAEFFER states that the percentage of breeze clinging to a unit weight of coke, is a function of the size of the pieces comprising such unit weight, and of the moisture present in both breeze and coke, and he prepared Table LXXXVIII as a result of his determinations in this regard.

The higher moisture content, usually present in domestic coke, causes difficult screening, and this difficulty can only be overcome by agitation, the violence of this agitation being dependent upon the size of the coke pieces, as well as upon the moisture content; if both the coke and the attendant breeze were dry while being screened, the clinging of the breeze would cause but little difficulty, but moisture must always be taken into consideration, and consequently all screening plants designed to handle domestic coke must be so designed as to violently agitate the coke pieces.

TABLE LXXXVIII.—BREEZE CLINGING TO COKE. SHAEFFER

Grade	Test		Moisture	Per cent. of breeze by weight clinging to coke	Grade	Test		Moisture	Per cent. of breeze by weight clinging to coke
Foundry..	Test 1	Coke	0.92	1.55	Range	Test 1	Coke	6.10	2.00
		Breeze	12.50				Breeze	12.50	
	Test 2	Coke	5.60	4.18		Test 2	Coke	8.10	8.40
		Breeze	12.50				Breeze	12.50	
	Test 3	Coke	6.00	6.80		Test 3	Coke	8.10	6.30
		Breeze	17.00				Breeze	17.00	
	Test 4	Coke	0.00	0.55		Test 4	Coke	0.00	1.36
		Breeze	17.00				Breeze	17.00	
Blast Furnace	Test 5	Coke	1.38	1.87	Chestnut	Test 1	Coke	8.30	3.00
		Breeze	12.50				Breeze	12.50	
	Test 6	Coke	9.30	5.00		Test 2	Coke	10.00	10.40
		Breeze	12.50				Breeze	12.50	
	Test 7	Coke	9.30	6.10		Test 3	Coke	10.00	6.90
		Breeze	17.00				Breeze	17.00	
	Test 8	Coke	0.00	0.64		Test 4	Coke	0.00	1.47
		Breeze	17.00				Breeze	17.00	
Egg......	Test 1	Coke	3.10	1.08	Pea....	Test 1	Coke	9.20	5.40
		Breeze	12.50				Breeze	12.50	
	Test 2	Coke	8.20	6.70		Test 2	Coke	10.10	12.90
		Breeze	12.50				Breeze	12.50	
	Test 3	Coke	8.20	7.90		Test 3	Coke	10.10	10.10
		Breeze	17.00				Breeze	17.00	
	Test 4	Coke	0.00	1.20		Test 4	Coke	0.00	4.60
		Breeze	17.00				Breeze	17.00	

The domestic screening plant in use at Milwaukee is shown in Fig. 134; the coke is delivered by a belt conveyor to the rotary screens, each 6 ft. in diameter by 37 ft. long, these screens delivering the separated coke to the pockets located below. A belt conveyor transports the coke from any of these pockets through a secondary screen, and thence to a box-car loader, this secondary screen being employed to remove any breeze which may have been produced in the bins.

The coke in Everett, Mass., was screened through rotaries measuring 6 ft. in diameter by 30 ft. long, the screens being inclined at 1.25 in. per foot, turning at the rate of 20 revolutions per minute. The main screen is provided with four different sized openings varying from 7/8 in. square to 2.5 in. square, the coke being delivered by a belt from the crushers to the upper end of the screen, or to the end which contains the smallest openings. This screen produces five sizes of coke, viz., breeze, nut, stove, egg, and furnace, the latter, and largest, coke passing out the end of the screen. The breeze coming from these screens is again screened in an auxiliary rotary, producing breeze and chestnut coke, while the breeze formed in the bins is removed by passing the coke from the bins over flat plate screens before the coke is loaded in cars or other vehicles.

The cost of keeping rotary screens in repair is quite a serious drawback to their use, as the abrasive action of the coke in sliding across the screen in two directions is very serious. This condition has led to the gradual adoption of reciprocating, or shaking screens, of which quite a number of designs are on the market. The design of a reciprocating screen requires accurate knowledge of the end to be achieved, as well as of the primary action of the screen itself, as, unless properly designed, the vibration imparted to the supporting structure will be very severe. This undue vibration requires that the screens shall be balanced, this being accomplished by setting the various screens of the plant over each other and so arranging their motion imparting cranks that they shall be set at either 120 or 90 degrees from each other.

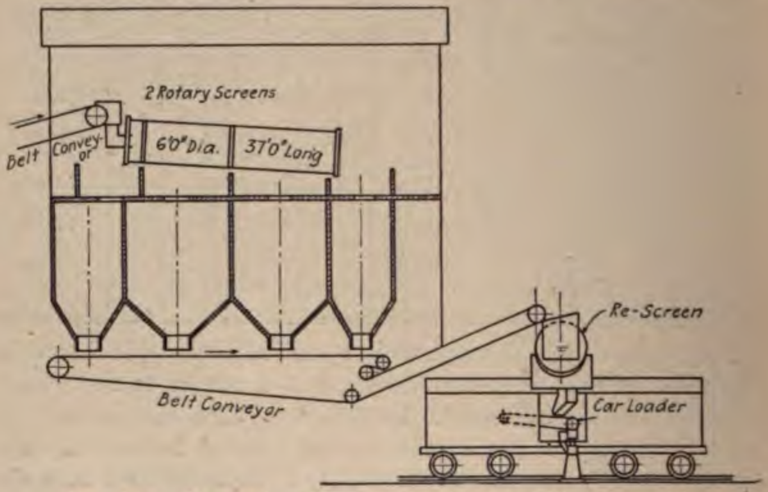

FIG. 134.—Domestic coke-screening plant at Milwaukee, Wis.

The new screening plant recently completed for the New England Gas and Coke Company, at Everett, Mass., is shown in Fig. 135; this plant consists of two complete units, each having a capacity of 75 tons of coke per hour, so arranged that both units, screening 150 tons, may be run at the same time, or either of them may be run independently. The entire plant is located in a building consisting of a heavy steel frame, the sides of the bins and the partitions being built of hard-burned red brick, with headers and facing of vitrified brick, all laid in cement mortar; all floors are of reenforced concrete construction, while the walls and roof above the bins, as well as the walls and roof of the bagging house located below the bins, are covered with corrugated asbestos-protected metal. The screens are of the shallow trough shaking type, consisting of two auxiliary screens 6 ft. wide and 11 ft. 6 in. long, used for feeding the crushers, two shaking conveyor screens of approximately the same dimensions, located under the crushers and above the main screens, and two main screens, each extending from the center of the building to the center of the last coke bin at each end of the structure, each of these two main screens being made in four sections 6 ft. wide and 11 ft. 6 in. long.

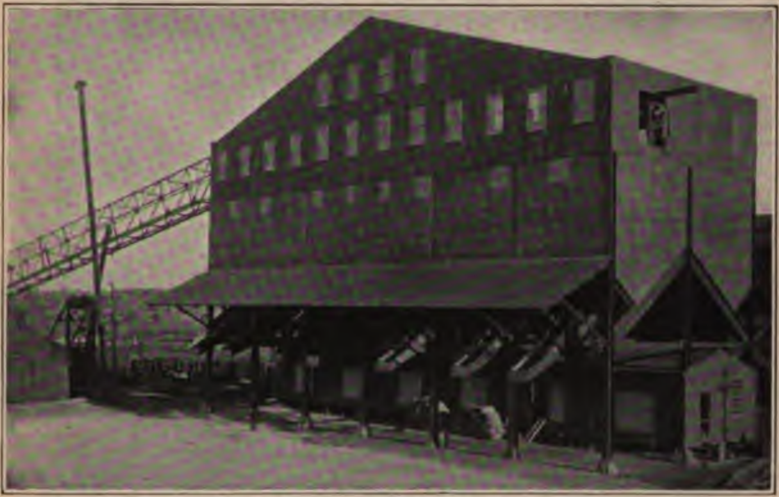

FIG. 135.—Coke-screening plant at Everett, Mass.

These screens are supported, or rather suspended, between two heavy steel girders, by means of pressed steel rocker arms, each screen being inclined slightly in the direction of the coke travel. The shaking motion is imparted to the screens by means of connecting rods fastened to the screen sections and driven by eccentric shafts, one of these eccentric shafts operating two screen sections. In order that vibration might be reduced to a minimum, the motion of each pair of screens is made reciprocating, and coiled steel springs are provided between each alternate rocker arm and the supporting girders, an even and regular motion being greatly assisted by the heavy fly-wheels on the eccentric shafts.

The shaking movement of the screens causes the coke to travel along the screen and, while sufficient shake is given to the mass to insure a thorough and efficient separation into the various sizes desired, this shake is not severe enough to break or fracture the lumps, or to produce any additional breeze. The breeze is delivered to a central storage bin, and nut, egg, and furnace coke

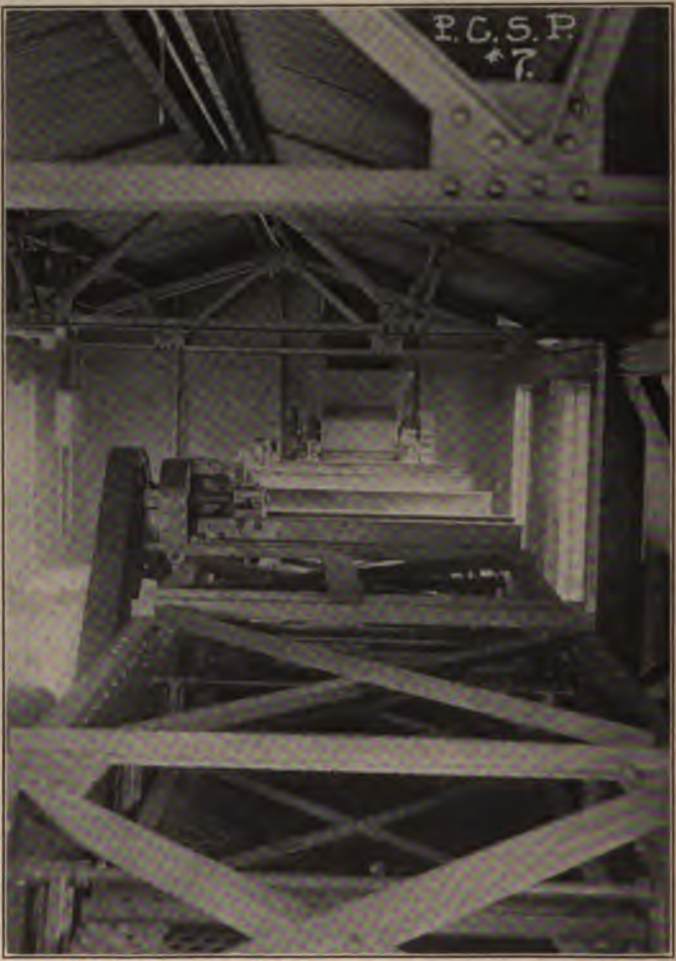

FIG. 136.—Detail of coke screen at Portland, Me.

is deposited in separate bins in the order named. The bottoms of the bins are provided with car and cart chutes, these chutes being hinged near the point of connection with the bin in such manner that they can be lowered when the car is in position to receive coke, the chute being counter-balanced to facilitate operation; valves are provided at the upper ends of these chutes for closing off the flow of coke when desired. Bagging chutes are also supplied for filling bags with nut, egg, or furnace coke, these chutes being provided with cut-off valves, operated by means of a foot-pedal, the coke passing into the bag being first conducted over a grizzly.

The car and cart chutes are each provided with stationary step screens for the purpose of separating from the coke any fines which may collect in the storage bins, these screens being set at an angle of 45 degrees. Breeze conveyors, consisting of 14-in. wide rubber belts, are provided under the stepped screens of the car and cart chutes, as well as under the grizzly of the bagging chutes; these conveyors deliver the breeze, or fines, to the boots of two breeze elevators located in the center of the building, which in turn deposit the fines in the breeze bins.

Fig. 136 shows a coke screen at Portland, Me., of similar design to that at Everett, except that its capacity is less; this figure shows the rigid construction necessary for this type of screen.

Another screen, which is used to some extent in Europe, is of the flat type, supported on each of the four corners by means of a ball-bearing standard; the movement of the screen is effected by the throw of a crank or eccentric located on a vertical shaft, this motion being imparted to the screen through the medium of flexible wire ropes.

In order to avoid the vibration produced by so many shaking screens, it has been proposed to suspend a flat screen by means of four wire ropes, one attached to each corner of the screen, and to give motion to the screen through the medium of a vertical shaft passing through a rigid bearing mounted on the screen frame, the frame receiving an eccentric motion due to an unbalanced fly-wheel keyed to the vertical shaft. This fly-wheel is provided with a movable weight, and the location of this weight on the wheel, or its position with respect to the center of the shaft and the speed of the latter, determines the degree of vibration given to the screen.

As stated before, the character of all coke is dependent entirely upon its method of production, and its usefulness will depend in great measure upon its nature, as hard coke will burn approximately as long as soft coke. As regards this degree of hardness, it is not necessary that metallurgical coke shall be hard enough to bear the load of a charge in a large smelting furnace, which **Simmersbach** estimates to be 42.6 lb. per square inch, and from his investigations he is led to believe that it is more important for the coke to be able to withstand frictional forces than crushing, because in smelting operations the particles which are rubbed off the coke by friction form thick masses which impede the proper working of the furnace. Under these conditions, coke which crumbles easily can be used only in small furnaces.

Simmersbach found that in screened coke the water content of the fine coke was considerably higher than that of the larger pieces. Another test[1] on the absorption of water by coke from two coking plants, I and II respectively, was made on pieces of medium size. This coke was thoroughly dried, weighed, and then immersed in water, being then reweighed at intervals as given below. The hygroscopic property was determined by placing thoroughly dried coke in a box in which cold steam was injected. The results of these tests showed the following water absorption:

Weighing interval, hours	Works I, per cent.	Works II, per cent.
1/4	6.20	4.2
1/2	7.23	4.6
1	7.70	5.2
2	9.10	6.6
6	11.20	8.7
12	12.90	9.9
24	14.10	10.9
72	17.20	13.5
144	17.30	13.6

The hygroscopic test showed:

8	0.22	0.36
12	0.33	0.38
24	0.33	0.38
48	0.37	0.30
72	[illegible]	[illegible]
96	[illegible]	[illegible]

The American Radiator Company ran a series of tests to determine the efficiency of domestic fuels using a 5-25-W Arco house-heating boiler for the purpose; this boiler contained 1150 sq. ft. of heating surface and 3.14 sq. ft. of grate surface, while the water surrounded the fire-pot and extended up through nipples into the intermediate and drum sections. The results of these tests on various fuels showed the following:

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[1] [illegible] Aug. 13, 1912, page 453.

Fuel	Pocahontas coal	Anthracite coal	Pittsburgh coke	Gas coke, egg	Gas coke, nut
B.t.u. per pound	14,750	12,679	12,183	12,183	12,183
B.t.u. obtained per pound	9,576	8,477	8,694	9,183	9,586
Efficiency, per cent	66.10	66.30	71.40	75.43	78.7
Per cent. of rated capacity	75.00	81.00	54.50	52.30	54.6
Cost per ton	$6.00	$7.10	$5.00	$5.00	$5.00
Cost of raising 1000 lb. of water 100°F	3.07¢	4.19¢	2.87¢	2.72¢	2.61¢
Cost in proportion to heating value	$4.40	$7.10	$3.40	$3.25	$3.12

An examination of these figures shows that, while the efficiency of the coke is greater than that of coal, the capacity of the boiler is less, this condition being explained by the greater bulk of the coke, and therefore shows that by installing a boiler with a larger fire-box it becomes possible to increase the capacity of the boiler until it equals that given with hard coal. WYMAN[1] states that the first essential in house-heating fuels is the rapidity with which a fire may be kindled and brought to a good heat; in this particular it was shown that Pocahontas coal was the easiest kindled of all the different fuels tested, gas coke of nut size being next, followed by gas coke of egg size, Pittsburgh coke, with anthracite coal last. The data obtained by these tests made it evident that gas coke possessed a decided advantage over other fuels for domestic purposes; it can be kindled easily, burns with an even, smokeless fire, is easily controlled and clean to handle; furthermore, it was shown to be the most economical fuel on the market, and there is no reason why it should not displace many forms of domestic fuel now in use.

A test reported by S. TAGG,[2] and made at an important pumping station in the vicinity of London, is of interest because gas coke was used as boiler fuel to determine its steaming efficiency

[1] J. H. WYMAN, Prog. Age, Oct. 1, 1909, page 765.

[2] Jnl. of Gas Ltg., Feb. 17, 1914, page 424.

as compared with smokeless Welsh coal. The boilers used were of the Cornish type, the coke being fired at from 30- to 40-minute intervals, while the coal was fired every 20 minutes, natural draft with a vacuum of 0.125 in. at the furnace doors being used.

The coke fire gave the following results:

Average steam pressure, pounds per square inch	101.5
Average temperature, °F	175.0
Average coke per square foot grate per hour, in pounds	12.7
Water evaporated from and at 212° per pound of coke, in pounds	10.37

Analyzing Coke.—The methods of analyzing coke are in the main the same as those given for coal. The method of determining the phosphorus in coke, as recommended by the American Foundrymen's Association, gives excellent results, and it is therefore explained here in detail.

Five grams of coke are ignited in either a platinum dish or large platinum crucible until all carbon is burned off, after which 10 cc. of dilute hydrochloric acid (1:1) and 20 cc. of hydrofluoric acid are added; the sample is then evaporated to dryness, and ignited at a dull-red heat. The residue is fused with about 3 grams of sodium carbonate and 0.3 gram of potassium nitrate, cooled, and the dish containing the fusion is placed in a beaker of water and boiled. When the fusion has been completely dissolved, the dish is rinsed into the solution and then laid aside. The solution is next acidified with hydrochloric acid, precipitated with ammonia, boiled, filtered, and washed with hot water. The precipitate is dissolved by washing the filter with warm dilute nitric acid, but if it should not dissolve under this treatment, it should be washed with warm dilute hydrochloric acid until it does dissolve. If the latter operation becomes necessary the solution must be evaporated to a volume of about 5 cc., after which 30 cc. of nitric acid (1.20 sp. gr.) is added; the solution is again evaporated to about 5 cc. and 30 cc. of nitric acid (1.20 sp. gr.) is again added. The solution is now heated to between 70° and 90°C. (158° and 194°F.), and 50 cc. of molybdate solution is added; this solution is stirred for a few minutes, filtered, and washed five times with a 3 per cent. nitric acid solution and five times with a 0.1 per cent. potassium nitrate solution. The precipitate and filter is now transferred to the flask in which the precipitate was made. Here 30 cc. of water are added, followed by

$\frac{N}{5}$ sodium hydroxide from a burette until the latter is in excess, the solution being constantly agitated. After the yellow precipitate has been entirely dissolved, 0.1 cc. of phenolphthalein solution is added as an indicator, and the result is then titrated with $\frac{N}{5}$ sulphuric acid. The following calculation will then give the percentage of phosphorus present:

$$\text{cc. } \frac{N}{5} \text{ NaOH} = \text{cc. } \frac{N}{5} \text{ H}_2\text{SO}_4 \times 0.0054 = \text{percentage of phosphorus.}$$

The molybdate solution is made by adding 100 grams of molybdic acid to 250 cc. of water, after which 150 cc. of ammonia is added; this mixture is stirred until all is dissolved, after which 65 cc. of nitric acid (1.42 sp. gr.) is added. Another solution is now made by adding 400 cc. of concentrated nitric acid to 1100 cc. of water; both solutions are allowed to cool, and the first solution is then slowly poured into the second, constantly stirring the mixture, after which a few drops of ammonium phosphate are added.

When apparent specific gravity determinations of coke are made, the sample should be sized so that all lumps are approximately of the same size and shape. When the sample is immersed, care should be taken to move the hydrometer rapidly up and down in the water several times, so that all air bubbles may be removed. Coke samples are very porous, in consequence of which they take up water very rapidly, and therefore they should not be permitted to remain in contact with the water for more than 5 minutes during any determination. Care should also be taken to see that all coke samples are thoroughly dry before any specific gravity determinations are made.

The Shatter Test.—The shatter test is used to determine the relative breakage of coke, and it usually consists of dropping a definite weight of coke, from a height of 6 ft., onto an iron plate. The apparatus adopted by the Bureau of Mines for making this test is shown in Fig. 137, and it consists of a box, having a capacity of 100 lb. of coke, mounted on a support, the bottom of the coke box being 6 ft. above the cast-iron shatter plate; the outer edges of this plate are protected by boards about 8 in. high in order that no coke may be lost. The bottom of the coke box is provided with hinged doors, these doors being held closed by means of a latch and so arranged that they will swing freely when the latch is released and the doors opened, thus permitting of a free fall for the coke. A sample of approximately 50 lb. of coke is placed in the box, and this sample is dropped four times on the cast-iron plate, the small material and dust being returned to the box with the large coke after each drop. After the coke has been dropped four times, it is screened on a 2-in. mesh screen, the coke passing through the screen and that remaining on it being weighed separately, thus determining the amount of breakage.

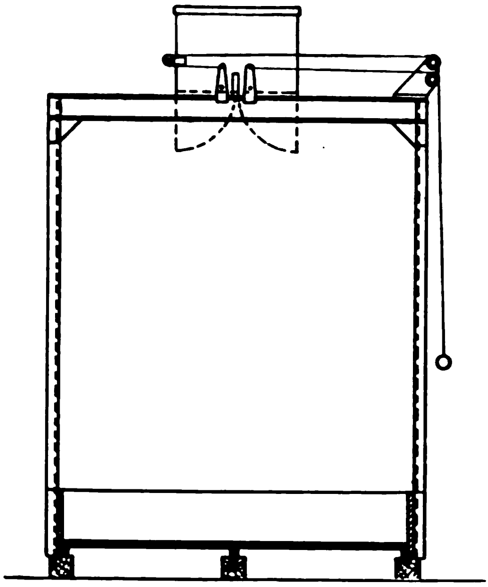


FIG. 137.—Apparatus for coke shatter test.

APPENDIX

HEAT UNITS

A French calorie = 1 kg. of water heated 1°C. at or near 4°C.
A British thermal unit = 1 lb. of water heated 1°F. at or near 32°F.
A pound-calorie unit = 1 lb. of water heated 1°C. at or near 4°C.
1 French calorie = 3.968 B.t.u. = 2.2046 pound-calorie.
1 B.t.u. = 0.252 French calorie = 0.555 pound-calorie.
1 pound-calorie = 1.8 B.t.u. = 0.45 French calorie.
1 B.t.u. = 0.252 French calorie = 0.555 pound-calorie.
1 pound-calorie = 1.8 B.t.u. = 0.45 French calorie.
1 B.t.u. = 778 foot-pounds = Joule's mechanical equivalent of heat.

CHEMICAL EQUATIONS FOR COMBUSTION IN OXYGEN

Hydrogen (H).
Equation.............. $2H_2 + O_2 = 2H_2O$
Relation by volume..... (2 vols.) + (1 vol.) = (2 vols.)
Relation by weight...... 1 + 8 = 9

Carbon Monoxide (CO).
Equation.............. $2CO + O_2 = 2CO_2$
Relation by volume..... (2 vols.) + (1 vol.) = (2 vols.)
Relation by weight...... 7 + 4 = 11

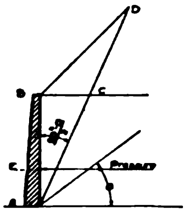
Ethylene, or Olefiant Gas (C_2H_4).
Equation.............. $C_2H_4 + 3O_2 = 2CO_2 + 2H_2O$
Relation by volume..... (1 vol.) + (3 vols.) = (2 vols.) + (2 vols.)
Relation by weight...... 7 + 24 = 22 + 9

Methane, or Marsh Gas (CH_4).
Equation.............. $CH_4 + 2O_2 = CO_2 + 2H_2O$
Relation by volume..... (1 vol.) + (2 vols.) = (1 vol.) + (2 vols.)
Relation by weight...... 4 + 16 = 11 + 9

HEATS OF COMBUSTION OF GASES IN OXYGEN. THOMPSON

Gas	Symbol	Products of combustion at 18°C. (64.4°F.), water liquid	Heat units evolved		Kilo-calories per cubic meter	B.t.u. per cu. ft.
			Calories per kg. of gas	B.t.u. per lb. of gas		
Acetylene....	C_2H_2	$2CO_2 + H_2O$	11,917	21,421	13,881	1,554
Benzene.....	C_6H_6	$6CO_2 + 2H_2O$	10,102	18,183	35,300	3,954
Carbon monoxide......	CO	CO_2	2,436	4,385	3,055	342
Ethane.......	C_2H_6	$2CO_2 + 3H_2O$	12,420	22,356	16,692	1,870
Ethylene.....	C_2H_4	$2CO_2 + 2H_2O$	11,931	21,476	14,967	1,677
Hydrogen....	H_2	H_2O	34,180	61,524	3,062	344
Methane.....	CH_4	$CO_2 + 2H_2O$	13,320	23,976	9,548	1,070

HORIZONTAL PRESSURES EXERTED AGAINST VERTICAL WALLS PER FOOT OF LENGTH



Anthracite coal = 52 lb. per cubic foot $a = 27°$
Bituminous coal = 50 lb. per cubic foot $a = 35°$
Retort coke = 35 lb. per cubic foot $a = 45°$
(20 per cent. moisture content)

$AE = \frac{AB}{3}$.

All pressures in pounds.

Depth in feet, *AB*	Anthracite coal				Bituminous coal				Retort coke			
	Horizontal surface *BC*		Surcharged *BD*		Horizontal surface *BC*		Surcharged *BD*		Horizontal surface *BC*		Surcharged *BD*	
	Total pressure	Pressure on lowest foot	Total pressure	Pressure on lowest foot	Total pressure	Pressure on lowest foot	Total pressure	Pressure on lowest foot	Total pressure	Pressure on lowest foot	Total pressure	Pressure on lowest foot
1	10	10	14	14	6	6	10	10	3	3	5	5
2	39	29	57	43	25	19	40	30	12	9	21	16
3	88	49	128	71	57	32	90	50	27	15	46	25
4	156	68	228	100	102	45	160	70	48	21	82	36
5	245	89	356	128	159	57	250	90	75	27	128	46
6	352	107	512	156	229	70	360	110	108	33	186	58
7	479	127	697	185	312	83	490	130	147	39	253	67
8	626	147	910	213	407	95	640	150	192	45	330	77
9	792	164	1152	242	516	109	810	170	243	51	418	88
10	978	186	1422	270	637	121	1000	190	300	57	513	97
11	1183	205	1721	298	770	133	1210	210	363	63	621	108
12	1408	225	2048	327	917	147	1440	230	432	69	739	118
13	1653	245	2403	355	1076	159	1690	250	507	75	867	128
14	1917	264	2787	384	1248	172	1960	270	588	81	1005	138
15	2201	284	3200	413	1433	185	2250	290	675	87	1153	148
16	2504	303	3640	440	1630	197	2560	310	768	93	1312	159
17	2826	322	4110	470	1840	210	2890	330	868	100	1482	170
18	3169	343	4607	497	2063	223	3240	350	973	105	1562	180
19	3531	362	5133	526	2298	235	3610	370	1089	109	1852	190
20	3912	381	5688	555	2548	250	4000	390	1200	118	2052	200
21	4313	401	6271	583	2809	261	4410	410	1323	123	2262	210
22	4734	421	6883	612	3083	274	4840	430	1451	128	2583	221
23	5174	440	7523	640	3369	286	5290	450	1586	135	2714	231
24	5633	459	8191	668	3669	300	5760	470	1728	142	2955	241
25	6113	480	8888	697	3981	312	6250	490	1875	147	3206	251

ANGLES OF REPOSE AND SLIDE OF COKE, IN DEGREES, ON SMOOTH SLABS. J. W. SHAEFFER, A. G. I., 1913

On concrete slabs

Coke	Foundry			Blast furnace			Egg			Range			Chestnut			Pea			Breese		
	Regular	Dry	Wet	Regular	Dry	Wet	Regular	Dry	Wet	Regular	Dry	Wet	Regular	Dry	Wet	Regular	Dry	Wet	Regular	Dry	Wet
Repose	19.7	19.7	20.8	20.5	20.3	20.7	20.0	20.3	20.8	22.1	22.2	20.8	22.9	22.8	23.9	30.8	30.8	31.9	34.2	33.1	35.2
Slide	20.7	20.7	21.8	21.6	21.3	21.7	20.9	21.3	21.8	23.1	23.2	21.8	23.9	23.8	24.9	31.8	31.8	32.9	35.2	34.1	36.2
Moisture	0.98	0.0	8.0	1.4	0.0	8–10	3.1	0.0	8–10	8.3	0.0	8–10	6.1	0.0	9–11	9.2	0.0	10–12	12.8	0.0	13–15
On cast-iron plate																					
Repose	19.2	19.7	19.3	20.8	20.8	20.7	19.2	19.1	19.7	21.1	20.6	19.9	21.9	21.8	22.7	23.8	23.4	30.7	30.7	31.0	34.9
Slide	20.2	20.7	20.3	21.8	21.8	21.7	20.2	20.1	20.7	22.1	21.6	20.9	22.9	22.8	23.7	24.8	24.4	31.7	31.7	32.0	35.9
Moisture	0.98	0.0	8.0	1.4	0.0	8–10	3.1	0.0	8–10	6.1	0.0	8–10	8.3	0.0	9–11	9.2	0.0	10–12	12.8	0.0	13–15
On steel plate																					
Repose	13.7	13.8	13.1	14.4	14.0	13.2	17.2	16.2	13.6	18.2	18.3	14.0	17.9	18.3	14.4	20.2	21.0	21.9	32.4	27.6	35.4
Slide	14.2	14.3	13.6	14.9	14.8	13.6	18.5	16.8	14.1	19.5	18.9	14.4	19.1	18.9	15.0	21.1	21.4	22.9	34.5	28.3	36.4
Moisture	0.98	0.0	8.0	1.4	0.0	8–10	3.1	0.0	8–10	6.1	0.0	8–10	8.3	0.0	9–11	9.2	0.0	10–12	12.8	0.0	13–15

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